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In Situ Leach Mining

Proceedings: Bureau of Mines Technology
Transfer Seminars, Phoenix, AZ, April 4,
and Salt Lake City, UT, April 6, 1989



UNITED STATES DEPARTMENT OF THE INTERIOR

Information Circular 9216

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By Staff, Bureau of Mines

**UNITED STATES DEPARTMENT OF THE INTERIOR
Manuel J. Lujan, Jr., Secretary**

**BUREAU OF MINES
T S Ary, Director**

PREFACE

In April 1989, the U.S. Bureau of Mines held technology transfer seminars on in situ leach mining at Phoenix, AZ, and Salt Lake City, UT. The papers presented at those seminars, as well as three reports of related research, are contained in this Information Circular. The papers highlight various aspects of Bureau research on in situ leach mining, including a field experiment in copper leaching in Arizona and the use of computer programs to model physical and chemical parameters involved in in situ mining.

The technology transfer seminar used as a forum for the transfer of this research is one of the many mechanisms used by the Bureau in its efforts to move research developments, technology, and information resulting from its programs into industrial practice and use. To learn more about the Bureau's technology transfer program and how it can be useful to you, please write or telephone:

U.S. Bureau of Mines
Office of Technology Transfer
2401 E Street, NW.
Washington, DC 20041
202-634-1224

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

atm	atmosphere, standard	L/min	liter per minute
°C	degree Celsius	M	molar
c/lb	cent per pound	m	meter
cm	centimeter	mD	millidarcy
cm ²	square centimeter	MHz	megahertz
cm ³	cubic centimeter	min	minute
cP	centipoise	mL	milliliter
D	darcy	mL/min	milliliter per minute
ft	foot	mm	millimeter
ft ²	square foot	μm	micrometer
ft/mi	foot per mile	μmho/cm	micromho per centimeter
ft/min	foot per minute	mm/min	millimeter per minute
ft/s	foot per second	mol	mole
g	gram	mol/L	mole per liter
gal/d	gallon per day	MPa	megapascal
(gal/d)/ft ²	gallon per day per square foot	ms	millisecond
gal/min	gallon per minute	mt	metric ton
g/cm ³	gram per cubic centimeter	mt/yr	metric ton per year
g/L	gram per liter	mV	millivolt
h	hour	pct	percent
Hz	hertz	ppm	part per million
in	inch	psi	pound (force) per square inch
Kbyte	kilobyte (1,024 bytes)	rpm	revolution per minute
kg	kilogram	s	second
km/s	kilometer per second	st/d	short ton per day
kW	kilowatt	st/yr	short ton per year
L	liter	wt pct	weight percent
lb	pound	yr	year
lb/st	pound per short ton	\$/yr	dollar per year
L/h	liter per hour		

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IN SITU LEACH MINING

**Proceedings: Bureau of Mines Technology Transfer Seminars,
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By Staff, Bureau of Mines

ABSTRACT

As part of its research on advanced mining technology, the U.S. Bureau of Mines emphasizes studies on in situ leach mining. This research enables the Bureau to develop technical information that can allow mining companies to adopt in situ leach mining methods. Research results should help to remove the technical and economic barriers currently hindering the development of commercial in situ leach mining operations.

This report contains papers summarizing results of Bureau research on various aspects of in situ leach mining. A featured topic is the cooperative field experiment of the Bureau and Santa Cruz Joint Venture in a copper deposit near Casa Grande, AZ. Discussions include an overview of the entire project, cost modeling, environmental permitting, laboratory core-leaching experiments, petrographic studies, and an evaluation of the geologic structure in the region. In addition, papers describe hydrologic modeling and geophysical studies of solution flow during leaching. Preliminary geochemical modeling studies are also summarized. A final paper describes a nonionic chemical additive with potential for significantly reducing the high costs of drilling in in situ leach mining.

INTRODUCTION

During the 1980's, the U.S. minerals industry found it difficult to compete with foreign mining companies in world markets. The U.S. copper companies were high-cost producers in a shrinking world market. Copper mining companies began restructuring in order to survive. The restructuring included considerable closures to match production with lower demand. Closing inefficient operations and adopting new processing technology enabled companies to boost productivity. This productivity, coupled with recent changes in the foreign exchange rates of the U.S. dollar, returned many U.S. copper producers to better fiscal health.

The present favorable market conditions will not, however, last forever. Major foreign companies also cut costs with considerable restructuring. U.S. companies remain high-cost producers. For continued existence, the U.S. companies must be able to sell mineral products at the same price as do foreign companies and still make a profit. Low ore grades, high labor rates, strict regulatory constraints, higher concern over miner health and safety, and more costly environmental safeguards all force U.S. mining costs upward. During the next market contraction, U.S. companies will again be particularly vulnerable. Most of the restructuring during the early 1980's was a one-time cure; similar actions will not likely be available to these companies in another down market. The Bureau of Mines believes that the industry needs new technology to overcome the high cost of conventional mining operations.

The Bureau recently expanded its efforts to provide technology that will help the minerals industry develop the necessary new mining and processing methods. Bold new methods of mining that improve worker productivity, minimize worker exposure to hazardous underground condi-

tions, profoundly alter environmental impacts, and reduce waste have moved to the forefront of Bureau research.

One of these new methods is called in situ leach mining. In situ leach mining involves circulating dilute chemical solutions through an ore deposit to dissolve the target metals. The solutions may be applied to the top of the ore body and allowed to seep down through it by gravity. Or the solutions may be forced through the ore between series of vertical injection and recovery wells (fig. 1). This method is sometimes referred to as "true" in situ leach mining. In yet another variation, leach solutions are trickled over rubble ore in stopes or other underground openings. Regardless of the application method, the solutions containing the dissolved metals are then pumped out of the ground and to a plant for metal recovery. After removing the metals from solution, operators recycle the solution back to the ore body. In situ leach mining thus combines mining and processing operations.

In situ leach mining offers several economic, safety, and environmental advantages over conventional mining. The Bureau believes that the method requires less capital investment. By eliminating ore extraction and crushing, in situ leach mining also saves labor and energy costs. Other benefits include fewer health and safety risks as miners will not be exposed to the hazards of underground mining. The technique disrupts the environment less than does conventional mining; in situ leach mining removes little from the ground except the target metals. It also leaves a mine site that is easily reclaimed to its original condition by capping wells and dismantling the processing plant.

The big question with the method is the metal recovery. Operators experienced low recovery from shallow rubble deposits that were leached in the 1970's. Bureau experiments,

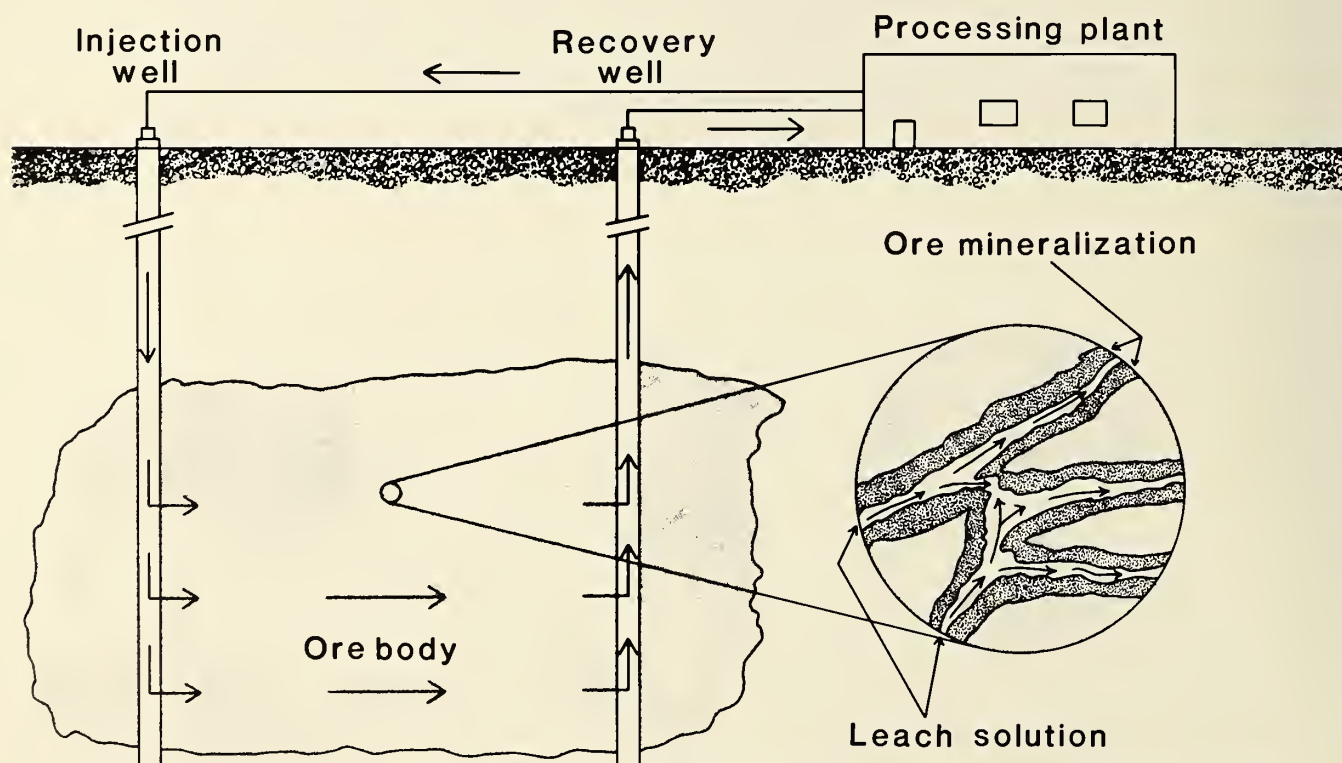


Figure 1.—Cross section of an in situ leach mining system in which the ore is leached via a system of vertical wells.

however, indicate that certain deposits should leach well. Such results underscore the necessity of conducting the tests and analyses described in this Information Circular before deciding whether to leach an ore body. The main environmental concern in in situ leach mining is the control and collection of leach solutions within the ore body. Developing control technology has been a major focus of the Bureau's research.

The overall advantages provide two very important benefits to mine operators, particularly to copper mining companies:

1. Production costs will be reduced, thereby making U.S. producers more competitive in the world market and reducing foreign import dependence.

2. Copper can be recovered from small and/or low-grade deposits that are currently not economical to mine by conventional methods.

The Bureau began studying methods of leaching ore left in the ground during the early 1970's. The early studies concentrated on fracturing shallow copper ore bodies to improve the flow of leaching solutions. By the late 1970's, the Bureau focused its in situ leaching research on uranium deposits in shallow, permeable sandstones. These studies helped stimulate commercial operations. Early in the 1980's, the Bureau began evaluating techniques for in-place leaching of other metals, such as manganese and gold. Column leaching experiments, geologic characterization, and fluid flow modeling predominated.

In 1986, the Bureau began a research program emphasizing in situ leach mining of shallow to moderately deep copper oxide ores. Its goal is to provide industry with the technology to design an in situ copper leach mining operation for any specific deposit. A combined laboratory and field testing effort was started to obtain site-specific data from two oxide copper deposits near Casa Grande, AZ. In support of the field work, the Bureau is conducting whole-core laboratory leaching experiments to determine the effects of microstructure, ore and gangue-mineral relationships, flow rates, and other factors on copper recovery and acid consumption.

The last Bureau of Mines technology transfer seminar on in situ leach mining technology was held in Denver, CO, in 1981. It emphasized uranium. This Information Circular contains the results of the Bureau's copper in situ mining research

over the past 5 yr and also includes the results of relevant copper block-cave leaching research.

The first five papers discuss several important aspects of the Bureau's cooperative in situ copper mining research near Casa Grande, AZ. The first presents an overview of the research. The second describes the leach mine design manual with cost model developed by Science Applications International Corp. under Bureau contract J0267001. The third explains the potential environmental impact of in situ copper leach mining and the requirements of regulatory agencies in Arizona. The fourth describes the laboratory tests necessary to evaluate copper deposits for leaching. The fifth paper discusses the potential effects of the geologic structures on leaching results at the deposit chosen for the experiment near Casa Grande, AZ.

The next paper discusses the potential applications of geochemical modeling for estimating chemical reactions that will occur during in situ leaching. The next two papers discuss the importance of hydrologic modeling studies in helping to understand the fluid flow during block-cave leaching. These papers describe the recently developed MINEFLO computer model and its usefulness in evaluating complex hydrologic problems.

The Bureau is also investigating potential applications of geophysical systems to monitor leachate during in situ mining. Tomographic reconstruction of seismic cross-hole data to help detect leach solutions is described in the next paper.

The final paper discusses the applicability of nonionic additives to improve drilling performance. Commercial in situ leach mining operators will spend much money on drilling wells. Successful application of this drilling additive will save considerable costs.

Additional technical information can be obtained by contacting William C. Larson, supervisor of the Advanced Mining Division at the Twin Cities Research Center, 5629 Minnehaha Avenue S., Minneapolis, MN 55417, or by calling 612-725-4690. Information concerning the Bureau's research programs on Advanced Mining Systems can be obtained by contacting Peter G. Chamberlain, program manager, 2401 E Street, NW., Washington, DC 20241, or by calling 202-634-9885.

IN SITU COPPER MINING FIELD RESEARCH PROJECT

By Jon K. Ahlness¹ and Daniel J. Millenacker²

ABSTRACT

To help meet the need of the domestic copper mining industry for new technology, the U.S. Bureau of Mines is conducting a field test of in situ copper mining at the Santa Cruz deposit near Casa Grande, AZ. A preliminary commercial mine design has been prepared for the site, based on a costing manual developed under Bureau contract. This paper describes the site and the well field design, as well as the steps involved in conducting the field test. These steps include obtaining geologic and hydrologic data on the ore zone, applying for environmental permits, constructing the well field and a solvent extraction-electrowinning plant, startup, leaching and processing operations, and well field decommissioning. Geologic characterization of drill core from the deposit is currently under way, along with whole-core laboratory leaching experiments. Test site geology and hydrology data will be gathered for development of final project designs and environmental permit applications.

INTRODUCTION

In recent years, copper production in the United States has experienced a significant decline; however, it is once again increasing, owing to the recent increase in copper price. For example, domestic production of copper in the years 1977-81 averaged 1.38 million mt/yr, while in the following 6 yr (1982-87) production averaged only 1.14 million mt/yr (a decrease of 18 pct) (1).³ Production in 1987 was up to 1.27 million mt, which is still down 8 pct from the 1977-81 average. Employment in the operating mines and mills fell from over 30,000 people in 1981 to less than 10,000 in 1987.

These decreases have occurred at a time when the United States is importing 27 pct of its copper requirements from Chile, Canada, Peru, Mexico, and other countries (2). Several factors are responsible for the drop in domestic production, including foreign competition, depletion of accessible, higher grade domestic reserves, and environmental considerations. Competition from developing countries is particularly intense in those cases where foreign reserves are higher grade, labor costs are low, and mines are nationalized or subsidized.

Maintaining a viable copper industry in the United States is dependent on several factors. First, it is up to the copper producers to maintain the lower operating costs and improved efficiencies that have been developed in recent years. Second, the development of new lower cost, environmentally sound technologies (such as in situ mining) are required to maintain and, it is hoped, to enhance domestic copper production from the remaining small, low-grade, and/or deep deposits. The

Bureau of Mines is conducting research to develop in situ mining technology to meet this need.

In situ mining has been used on a commercial basis since the mid-1970's to produce uranium from porous sandstone deposits in Texas and Wyoming (3). In situ methods have also been used to recover copper from low-grade ore and waste rock in old open pit mines, block-caved zones, and backfilled stopes as an afterthought to conventional mining (4). Ore can also be placed on an impermeable surface in heaps and leached by sprinkling a leach solution (lixiviant) over the surface of the pile. Heap leaching is actively being used for extracting gold and silver from low-grade ore (5). Recovery of the pregnant leach solution in each of these operations occurs by fluid movement to a recovery well or underground sump, or to a surface collection pond.

The Bureau's research objective is to stimulate domestic production of copper by the private sector, using in situ mining methods. To achieve this goal, the Bureau intends to provide industry with the means to design the most economical in situ mining operation for any specific copper oxide ore deposit. A "Draft Generic In Situ Copper Mine Design Manual" has been prepared by Science Applications International Corp. (SAIC), McLean, VA, under Bureau contract J0267001 (6). Subcontractors that assisted SAIC included Ray V. Huff and Associates, Golden, CO, Davy McKee Corp., San Ramon, CA, and Sergeant, Hauskins, and Beckwith, Phoenix, AZ. This draft manual serves as a source document for developing a commercial-scale design for an in situ operation. The draft manual contains

1. A listing and description of the design elements and procedures for each component of an in situ copper mining system, and a method of costing individual components.

2. A method of identifying the best of 42 possible in situ mining scenarios, which maximizes the discounted-cash-flow

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³Italic numbers in parentheses refer to items in the list of references at the end of this paper.

rate of return (DCFROR) for commercial operation for a specific site.

3. A listing of the site-specific parameters that must be quantified to develop a commercial design and a description of laboratory and field tests to measure these parameters.

4. A cash-flow model and computer program to conduct a DCFROR economic analysis incorporating all capital and operating expenses associated with the well field, a solvent extraction-electrowinning plant (SX-EW), and environmental permitting.

5. A description of the environmental procedures, specifications, designs, and costs for permitting, monitoring, and restoration.

The generic approach provides flexibility in developing designs and costs for a wide range of deposit characteristics and operational parameters. To verify the draft manual and demonstrate the technical feasibility of in situ copper extraction, a field test is being conducted in a copper oxide ore deposit in Arizona.

IN SITU COPPER MINING FIELD RESEARCH PROJECT

The Bureau is conducting the field research project in cooperation with the Santa Cruz Joint Venture (a mining partnership formed between ASARCO Santa Cruz Inc., a subsidiary of ASARCO Incorporated, and Freeport Copper Co., a subsidiary of Freeport-McMoRan Gold Co.). This research is being conducted at the Santa Cruz site, 7 miles west of Casa Grande, AZ. A commercial mine design has been prepared for this site using the algorithms and model provided in the "Draft Generic In Situ Copper Mine Design Manual" (6). Actual field testing will utilize the unit-cell concept. A unit cell is a small well field constructed to commercial-scale well size and spacing specifications. Since the design and operational parameters are comparable to a commercial-scale operation, the field test will allow validation of the algorithms used in preparation of the commercial design and will also demonstrate whether an in situ

facility can be operated as designed and within budget for a sustained period of time. Validation of the model will consist of comparison of actual versus calculated construction and operating costs, as well as verification of technical parameters (flow rate, fluid control, etc.).

The Santa Cruz site (fig. 1) is undeveloped and contains a mineralized zone with estimated copper oxide reserves of 97 million mt at an average grade of 0.7 pct acid-soluble copper. Chrysocolla and atacamite are the predominant copper oxides present in the granite and porphyry host rocks. Available field data indicate an in situ permeability range of 5 to 20 mD. The bottom of the mineralized zone is located at an average depth of 2,200 ft below the surface, with an average thickness of 345 ft. Surface terrain is essentially flat lying with relief of 10 ft/mi. Access to the site is provided by well-maintained State



Figure 1.—Drill rig on the Santa Cruz site.

and township roadways. Geologic characterization of available drill core is presently under way, along with laboratory whole-core leaching experiments.

A well field of two connected five-spot patterns has been selected as the design basis for the field test. Corner wells in the pattern will serve production needs, and center wells will be used for injection. Producer-to-producer well spacing is designed to be 160 ft. The selected well spacing and calculated flow rate allow a sufficiently large area to be leached to determine copper recovery, copper loading, and residence time expected for a commercial operation in a representative geologic environment. Short-radius, propped hydraulic fractures may be necessary to increase the effective wellbore radius.

The entire field test is scheduled to run for a 48-month period. Actual fluid injection-recovery operations will, however, run for only 18 months. Construction and decommissioning activities account for the balance. The 18-month production period will allow sufficient time to assess operational problems and to determine if copper loadings can be maintained over an extended period.

Several steps are involved in conducting the field test. The first of these is to obtain hydrologic and geologic data from the ore zone and vicinity. A test program will be initiated to assess, first, flow characteristics within the selected ore block and, subsequently, flow control within a well pattern. Initially, two holes will be drilled to obtain core and to allow geophysical logging and open hole testing. Testing will provide data on porosity and permeability variations within the vertical ore interval. Upon completion of testing, two additional wells will be completed within 50 ft of the original wells. Tests will be conducted to determine orientation of the natural fractures (and effectiveness of wellbore stimulation) and to assess fluid movement and control within the well pattern. These data will be used for orienting the five-spot patterns.

Environmental permitting activities will be initiated during the site investigation stage. Required ground water protec-

tion permit applications will be prepared and submitted to the appropriate Federal and State regulatory agencies. Permitting is the mechanism used by the regulatory agencies to ensure that applicable water quality standards will be achieved in the areas adjacent to the facility during both operation and decommissioning. Requirements specific to protection of air resources will additionally need to be addressed at the local government level. Ground water protection requirements applicable to an in situ mining operation in Arizona have previously been discussed by Weeks (7).

Upon completion of site characterization work, the well field will be constructed. The preliminary well field design assumes a 50-pct copper recovery over an approximate leach interval of 325 ft. The maximum flow rate expected for each of the injection wells is 40 gal/min. The pregnant solution grade is estimated to be 10 g/L. The final design (and parameter values) may be modified as field data become available from the site investigation.

A solvent extraction-electrowinning (SX-EW) plant will be constructed upon completion of, or in conjunction with, well field installation. The plant will extract copper from the pregnant liquor, plate it as cathode copper, and recondition the lixiviant. The facility design will achieve an anticipated copper production rate of 1,095 st/yr cathode copper.

Startup operations will initially occur over a 2-month period to establish flow rate equilibrium, determine steady-state solution concentrations, and identify and troubleshoot equipment problems. Upon completion of well field startup, the lixiviant will be injected and actual leaching operations will commence. Produced fluids will be pumped to the SX-EW plant for processing.

Well field decommissioning will be initiated at the completion of leaching. During this time, the surface plant will be dismantled and containment structures will be removed from the site. Wells will be abandoned and the leach interval restored as required by the regulatory agencies.

SUMMARY

The Bureau is conducting research to evaluate in situ mining as a method to maximize the probability of the domestic production of copper from small, deep, and/or low-grade copper oxide ore deposits. The Bureau intends to provide the U.S. copper mining industry with the technical and economic data necessary to design an in situ mining operation for any specific copper oxide ore deposit. A "Draft Generic In Situ Copper Mine Design Manual," prepared for the Bureau by Science Ap-

plications International Corp., serves as a source document for developing a commercial design for an in situ operation. To verify the draft manual and demonstrate the technical feasibility of in situ extraction of copper, a field test is being conducted in a copper oxide ore deposit in Arizona. Field research is expected to continue for the next several years. Research results will be made available to the mining community as significant milestones are attained.

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GENERIC DESIGN MANUAL: COST MODEL FOR IN SITU COPPER MINING

By Joseph M. Pugliese¹

ABSTRACT

The U.S. Bureau of Mines has provided a cost model for in situ copper mining that specifies (1) site-specific parameters, which must be quantified, (2) a method for individual site design based on these site parameters, and (3) a procedure for assessing the economic viability of the site-specific design. This paper describes the model, a menu-driven computer program that performs calculations for developing commercial design specifications as well as capital and operating costs. The program also provides discounted-cash-flow rate of return (DCFRR) and sensitivity analyses for a true in situ copper mining operation at any specified site. This model can be used to compare the relative costs of different in situ mining scenarios. It provides a systematic method for assessing the commercial feasibility of applying true in situ copper mining at a selected site.

INTRODUCTION

The Bureau of Mines believes that the competitive position of domestically produced copper can be significantly improved with the application of in situ mining techniques. A long-term objective of the Bureau is to increase the probability of the domestic production of copper by the private sector, using in situ means. As part of the effort to meet this objective, the Bureau conducted research on cost modeling of in situ copper mining to provide the mining industry with the means to design the most economically successful in situ copper operation for any specific deposit. The research was conducted by Science Applications International Corp. (SAIC), McLean, VA, under Bureau contract J0267001. The contractor had three subcontractors to aid in conducting the work: Ray V. Huff and Associates, Inc., Golden, CO; Davy McKee Corp., San Ramon, CA; and Sargent, Hauskins, and Beckwith, Phoenix, AZ.

The contract effort started in October 1986, and the contractor's final report was completed in April 1988.²

The paper deals with the economic analysis found in Volume II of SAIC's final report and the commercial-scale operation example found in Volume IV of that report. Mention of these volumes and of SAIC's final report throughout the paper will be made with the understanding that the work is cited in footnote 2.

The algorithms developed from this research are combined into the cost-modeling computer program for the design and economic analysis of an in situ copper mining operation. The computer program is on an IBM-compatible microcomputer; the minimum hardware requirements for running the program are a 360-Kbyte memory and a math coprocessor.

COST MODEL ASPECTS

The cost model may be used to evaluate 42 in situ mining scenarios. Three basic characteristics are used to define each of the 42 scenarios.

The first characteristic addresses the way in which the ore body and adjacent rock will be penetrated, that is, the method of deposit access. Seven types of deposit access treatments are—

1. Drilling from the surface;
2. Drilling from existing underground entries;
3. Developing new underground entries, with drilling from those entries;
4. Combinations of types 1 and 2;
5. Combinations of types 1 and 3;
6. Combinations of types 2 and 3; and
7. Combinations of types 1, 2, and 3.

The second characteristic involves the treatment of the rock matrix from within the borehole, or matrix modification. The three types of treatments considered by the cost model are—

- A. No modification;
- B. Modification using hydraulic fracturing; and
- C. Modification using explosives in the borehole.

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²Davidson, D. H., R. V. Huff, R. E. Weeks, and J. F. Edwards. Generic In Situ Copper Mine Design Manual (contract J0267001, Science Applications Int. Corp.). Volume I: Executive Summary, Apr. 1988, 93 pp.; Volume II: Draft Generic In Situ Copper Mine Design Manual, Apr. 1988, 454 pp.; Volume III: Lakeshore Field Experiment and Design of Commercial Scale Operation, Apr. 1988, 371 pp.; Volume IV: Santa Cruz Field Experiment and Design of Commercial Scale Operation, Apr. 1988, 385 pp.; Volume V: Field Testing at the Santa Cruz Site, Apr. 1988, 114 pp.; for inf., contact J. K. Ahlness, TPO, Twin Cities Res. Cent., BuMines, Minneapolis, MN.

The third characteristic involves the initial hydrologic setting of the ore body. Two types of treatments are—

- i. No hydrologic modification (already saturated); and
- ii. Hydrologic modification (unsaturated conditions originally).

Type ii treatment indicates that the ore body should be saturated before leaching begins.

Forty-two mining scenario permutations can be determined from the types of treatments shown above. An example of a mining scenario permutation would be 1.A.i., that is, drilling from the surface, no matrix modification, and an ore body that is initially saturated. The approach used to estimate which of the 42 possible mining scenarios may be best for a specific site is found in Volume II, chapter 9, of SAIC's final report.

The cost model does the following:

1. Develops detailed capital and operating costs for any proposed copper in situ mining project.
2. Solves for pretax discounted-cash-flow rate of return (DCFRROR) or selling price.
3. Contains cost data (default values) from a broad experience base.
4. Allows for cost analyses where either minimum or maximum data are available.
5. Accepts user-specified information.
6. Conducts cash-flow analyses.
7. Can be used for sensitivity analyses.

Each of these features will be covered in the following sections.

COST MODEL INPUT PARAMETERS

Input parameters are listed under the following categories: (1) business-related parameters, (2) site-specific ore body and well field characteristics, (3) copper leaching, (4) program control, (5) well system specifications and costs, (6) surface plant specifications and costs, and (7) environmental costs. The first three categories contain those input parameters that must be utilized to specify a certain mining scenario. The fourth category contains the input parameters that must be specified to control the printout of output data. The last three categories contain the input parameters that correspond to cost values and design specifications that translate a specific mining scenario into capital and operating costs.

Category 1 consists of inflation indexes for plant and mining construction costs, copper selling price or rate of return, capital expenditure schedule, plant life, and copper production.

Category 2 contains site-specific parameters that need to be considered in design of the wells and well pattern, such as depth to the bottom of the ore zone, thickness of the ore zone, leach interval, ore grade, permeability and porosity of the ore zone, type of mining access (surface or underground), status of underground mine access (existing or nonexisting), well

field flow pattern, well spacing, type of well completion and stimulation, and ore body water saturation.

Category 3 includes site-specific parameters that relate to pregnant liquor copper loading, acid consumption by gangue and copper minerals, and copper recovery efficiency.

Category 4 contains the parameters that control whether to print out calculated variables, annual cash-flow summary, and/or all input parameters.

Category 5 includes unit costs used to convert well system input parameters to output specifications and costs. Examples of these parameters are the cost of preparing the surface drill site; wellhead, packer, tubing, and casing costs; and hydraulic fracturing and explosive stimulation costs.

Category 6 includes unit costs used to compute surface plant specifications and costs. These unit costs apply to such input parameters as pipes (lines), extractant, and diluent. Flow rate through the lines is covered under this category.

Category 7 addresses environmental aspects of the in situ mining operation. Included here are monitoring well unit cost, cost of initial environmental permitting, annual cost of environmental monitoring, and well field restoration cost.

COST MODEL OUTPUT PARAMETERS

Output parameters related to the well system, the surface plant, and the environment are grouped under the following categories: (1) design variables, (2) capital costs, and (3) operating costs.

Category 1 includes leach area of unit cell; area of well field; injection pressure at top of ore zone; injection flow rate; total system flow rate; number of unit cells; well field life; injection acid concentration; acid and lime consumption; metal sulfates produced per year; number of injection and total wells (excluding monitor wells); well field solution transfer piping length and diameter; well-field-to-plant transfer piping diameter; total weight of copper in unit cell; fraction of copper recovered from the zone being leached; number of mine and

plant operators; tubing, casing, and wellhole diameters; and injection and production pump power requirements.

Category 2 includes well site preparation cost; well casing, cementing, drilling, completion, and stimulation costs; well logging cost; well injection and production equipment costs; cost per fan for fan wells; total surface facility capital cost; total cost of the solution transfer system; total well field capital cost; cost of initial environmental permitting; cost of monitoring wells; and well field restoration cost.

Category 3 includes chemical and consumable costs, labor cost, cost of utilities, cost of environmental monitoring, and maintenance costs.

CASH-FLOW ANALYSIS

This cost model can be used to conduct a cash-flow analysis, using the DCFROR. A thorough discussion on DCFROR is found in Stermole,³ and a brief summary from that reference is presented in this section. The term "discount" is generally considered to be synonymous with "present worth" in economic evaluation work. The term "cash flow" is used to refer to the net inflow or outflow of money that occurs during a specified operating period such as a month or a year. The term "discounted cash flow" evolved from the idea that investors often handle the time value of money using discounted (present value) calculations. That is, in evaluating the mining operation's economic potential, the investors used positive and negative cash flows at their present worth anticipated from an investment.

According to the Stermole text, the investment cash flow in any one year represents the net difference between inflows of money from all sources minus investment outflows of money from all sources. This cash flow is calculated by

$$cf_n = R - CC - OC,$$

where cf_n = net cash flow in each project year, \$/yr,

R = revenue, \$/yr,

CC = capital costs, \$/yr,

and OC = operating costs, \$/yr.

The cost model developed by SAIC for the Bureau does not consider taxes, royalties, or predevelopment costs in the calculations because of the generic nature of the cost model and the variation in these costs among States and companies. In addition, the model does not include sunk costs, such as for exploration and property acquisition; depletion allowances; depreciation; and salvage value. The cost model should be used to make relative production cost comparisons among different commercial in situ copper mining scenarios for the purpose of selecting the one most economical. The cost model can form the basis for a mining company to develop its own absolute cost model.

In using DCFROR, the term "net present value" (NPV) needs to be defined.⁴ The NPV is the cumulative present worth of positive and negative investment cash flow using a specified discount rate to handle the time value of money. As an equation:

$$NPV = \sum_{n=0}^N \frac{cf_n}{(1+i)^n},$$

where NPV = net present value in year of calculation, \$/yr,

cf_n = net cash flow in each project year, \$/yr,

n = year number (first year number = 0),

N = total project life (plant life + construction period), yr,

and i = discount rate, pct (expressed as a decimal in equation).

The DCFROR is the value of " i " that makes NPV equal to zero. In the cost model, the DCFROR is used to calculate cop-

per selling price if DCFROR is specified. Conversely, copper selling price is used to calculate DCFROR if copper selling price is specified.

The information required to define a commercial in situ copper mining operation for the Santa Cruz site near Casa Grande, AZ, for which design specifications and costs can be developed is found in Volume IV of the contractor's final report. In September 1986, the Bureau and the Santa Cruz Joint Venture (ASARCO Santa Cruz Inc. and Freeport Copper Co.) agreed to cooperate for the purpose of conducting data acquisition field tests and developing in situ mine designs for the Santa Cruz site. In May 1987, the Bureau contractor, SAIC, met with Asarco staff and obtained business decision and ore deposit information for use in defining a commercial-scale operation. In defining this operation, a scoping analysis was employed that did not use site-specific construction data pertaining to local Casa Grande conditions and did not consider local taxes and utility costs.

Based on SAIC's analysis, the initial mining scenario (best from information available at the time) involved vertical wells drilled from the surface. Vertical wells seemed to be a reasonable place to start since no underground workings existed. For this Santa Cruz commercial-scale operation example, a five-spot pattern with a radial flow pattern to be used for leaching was developed (fig. 1). In this figure, the central injection well region is an expanded view showing the wellbore radius prior to small-radius hydraulic fracturing and the radius after this type of stimulation. A value of 8 ft is assumed as the effective wellbore radius, or radius after stimulation. The four producer wells are at the corners of the pattern and have a spacing (S) of 160 ft. The depth to the bottom of the ore zone is 2,136 ft, and the ore zone thickness is 345 ft. The ore body is in a saturated state. The copper production rate (plant capacity) is estimated at 23,600 st/yr, and the plant life is 15 yr. The grade of ore is 0.72 pct, and the permeability of the ore zone is estimated at 2 mD. Underground copper recovery is assumed at 50 pct, and the copper loading is estimated to be 10 g/L. The other input and calculated parameter values for this Santa Cruz commercial-scale example can be found in Volume IV of SAIC's final report.

Table 1 shows the annual cash flows for the commercial-scale copper in situ mining operation example at the Santa Cruz site and is taken from Volume IV of SAIC's final report. The DCFROR is 20 pct, which resulted in a copper selling price of 51 (50.94) c/lb. The project life is 18 yr, including the first 3 yr for construction, which results in a plant life of 15 yr. As well fields are depleted of copper, they are replaced with new well fields. In this example, well field construction is estimated to take 1½ yr, and the well field life is 3 yr. As each well field is put into operation, a well priming time is required. For the years 4, 7, 10, 13, and 16, the lower production values shown reflect this priming effect. That is, each time a new well field is started up, no salable cathode copper is assumed to be available until 1 pore volume displacement has taken place. This lost production of cathode copper is subtracted from the normal annual production. The total operating and maintenance costs are incurred from the first year of production (year 4) through the last year of project life (year 18).

³Stermole, F. J., and J. M. Stermole. Economic Evaluation and Investment Decision Methods. Investment Evaluations Corp., Golden, CO, 1987, 479 pp.

⁴Work cited in footnote 3.

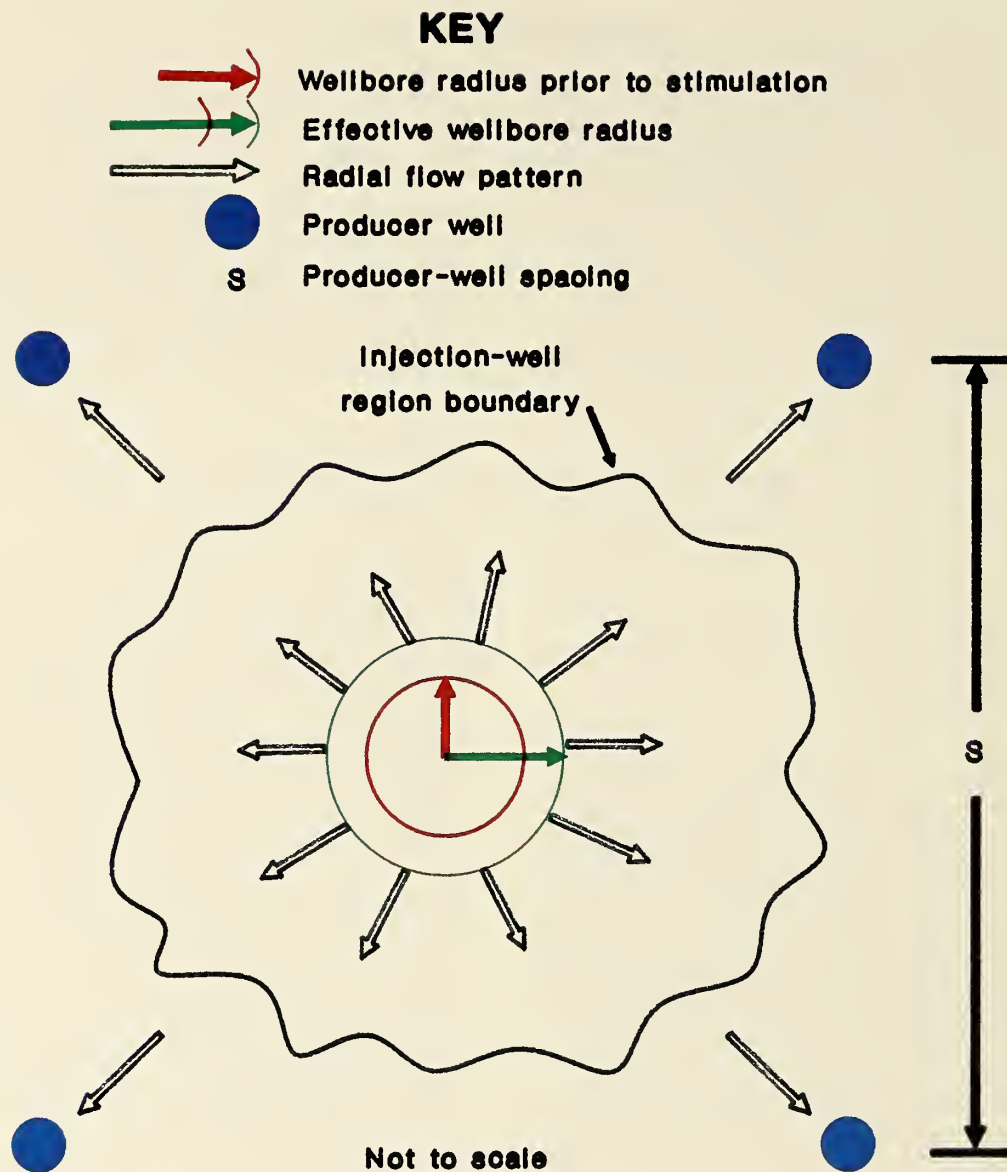


Figure 1.—Plan view of five-spot pattern, commercial-scale operation example.

Table 1.—Annual cash flow, Santa Cruz commercial-scale operation example

Year	Approximate production ¹ (Y), 10 ³ st/yr	Production revenue ² (R), 10 ³ \$/yr	Costs, 10 ³ \$/yr			Net cash flow (cf _n), 10 ³ \$/yr	Discounted cash flow ³ (DCF), 10 ³ \$/yr
			Plant capital cost (PCC)	Well field capital cost (WCC)	Total operating and maintenance cost (TOC)		
1	0	0	4,945	0	0	-4,945	-4,945
2	0	0	5,374	2,187	0	-7,561	-6,301
3	0	0	11,179	4,245	0	-15,424	-10,711
4	18	18,286	0	0	11,908	6,378	3,691
5	22	21,928	0	2,217	11,908	7,803	3,763
6	22	21,928	0	4,304	11,908	5,716	2,297
7	18	18,286	0	0	11,908	6,378	2,136
8	22	21,928	0	2,217	11,908	7,803	2,178
9	22	21,928	0	4,304	11,908	5,716	1,329
10	18	18,286	0	0	11,908	6,378	1,236
11	22	21,928	0	2,217	11,908	7,803	1,260
12	22	21,928	0	4,304	11,908	5,716	769.3
13	18	18,286	0	0	11,908	6,378	715.3
14	22	21,928	0	2,217	11,908	7,803	729.3
15	22	21,928	0	4,304	11,908	5,716	445.2
16	18	18,286	0	0	11,908	6,378	414.0
17	22	21,928	0	0	11,908	10,020	542.0
18	22	21,928	0	0	11,908	10,020	451.6

¹ Actual production: priming years, 17,949 st/yr; nonpriming years, 21,523 st/yr. On-stream plant operating time per year, 333 days (91.2 pct).

² Production revenue = actual production × 2,000 lb/st × selling price. Selling price is 51 (50.94) c/lb, and DCFROR is 20 pct.

³ Sum of these values = net present value (NPV) = 0.

NOTE.—Underground capital cost (UCC) is 0 \$/yr for life of project.

Source: Davidson, D. H., R. V. Huff, R. E. Weeks, and J. F. Edwards. Generic In Situ Copper Mine Design Manual (contract J0267001, Science Applications Int. Corp.). Volume IV: Santa Cruz Field Experiment and Design of Commercial Scale Operation, Apr. 1988, 385 pp.

SENSITIVITY ANALYSIS

This cost model may be used to conduct sensitivity analyses. Sensitivity analyses can be used to evaluate the effects of uncertainty on investment by determining how the investment profitability varies as parameters are varied one at a time.⁵ Further, sensitivity analyses may be used to identify those critical variables that, if changed, could affect substantially the profitability measure, such as DCFROR. When carrying out a sensitivity analysis, the value for an individual variable is changed, and the effect of the change on the expected rate of return (or other profitability measure) is calculated. Once the strategic variables that have major effects on the profitability measure are identified, they can be given special attention by the mine management. Several examples of parameters that may be allowed to vary in an in situ mining sensitivity analysis are copper loading, ore grade, well completion and type of stimulation, and well spacing. These may be plotted against selling price while holding DCFROR constant or against DCFROR while holding the selling price constant.

According to the Stermole book, the range approach to sensitivity analysis involves estimating the most optimistic and most pessimistic values (or the best and worst cases) for each parameter in addition to estimating most expected values. The best, worst, and most expected case sensitivity analysis results provide important information that bracket the range of project rate of return results that can reasonably be expected.

Figures 2 and 3 represent examples of sensitivity plots for the Santa Cruz test site with data taken from Volume IV of SAIC's final report. In using sensitivity analysis, different values are assigned to one input parameter, plotted against the x-axis, while other input parameters are held constant. As different values are given to the one input parameter, the corresponding changes in selling price are plotted against the y-axis. The selling price values are determined through the cash-flow analysis. For these sensitivity analysis examples, the DCFROR is 20 pct. The vertical line within each plot indicates the initial case value estimate for the parameter to be changed.

In figures 2 and 3, the initial case parametric value provides a selling price of 51 c/lb for the DCFROR of 20 pct. In general, in all of the examples an increase in the parametric value decreases the selling price. No estimates of ore grade above the initial case value of 0.72 pct were made (fig. 2C) because this value was thought to be at the high end for the Santa Cruz example.

Figure 3, the plot of selling price versus permeability, is an illustration of combined effects in the cash-flow analysis. The permeability increases reduce the number of wells and thus the total cost of one well field. By maintaining a fixed well spacing, the well field will be depleted of copper in less time because of the higher injection flow rate. (More well fields will be needed over the project life, which will cause additional capital costs.) The higher flow rate (higher permeability) can result in higher negative discounted cash flow if the initial well field life is less than 3 yr, thus requiring a higher selling price to maintain the 20-pct DCFROR.

⁵ Work cited in footnote 3.

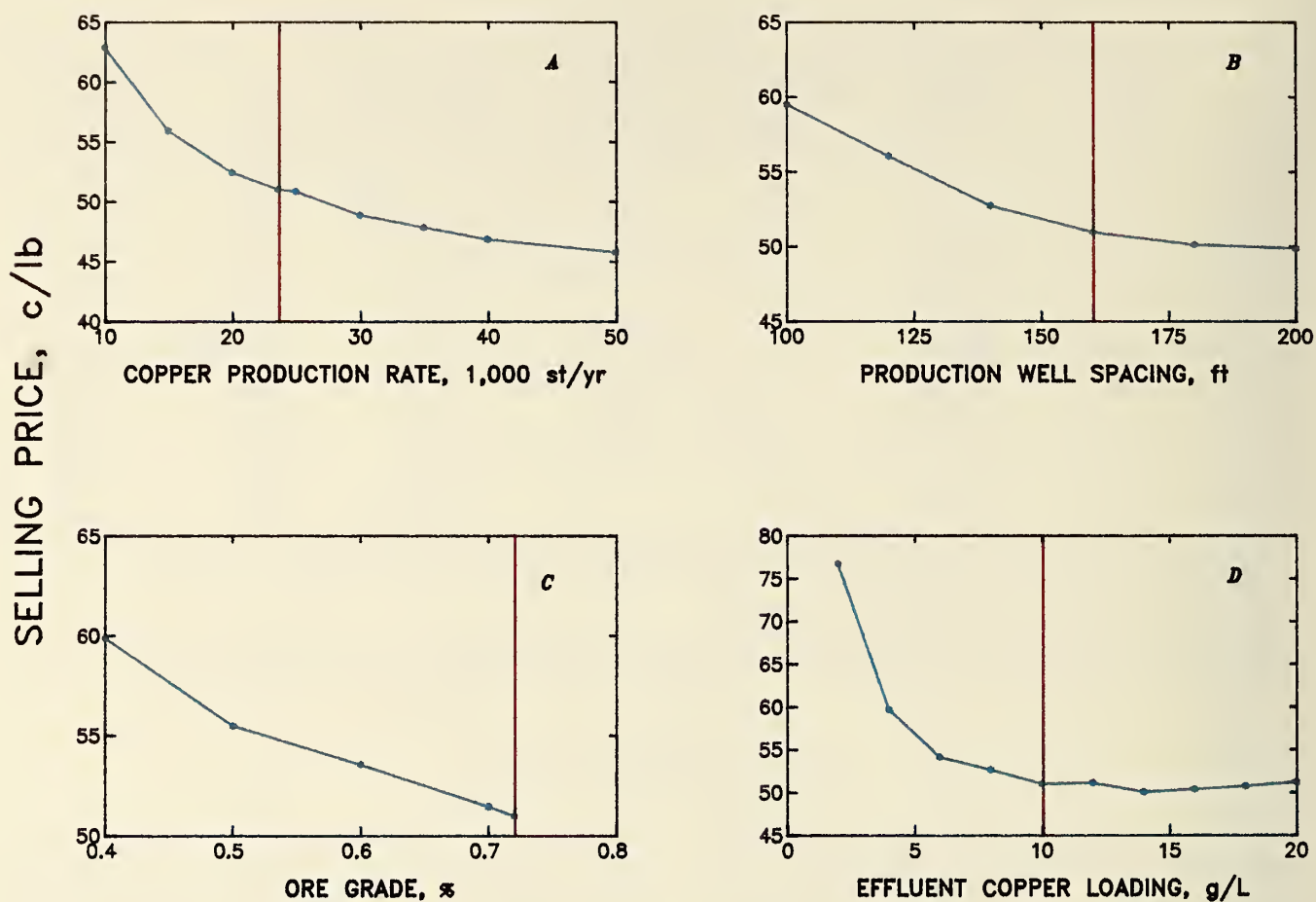


Figure 2.—Sensitivity analyses, Santa Cruz commercial-scale operation example. (Vertical line represents initial case.)

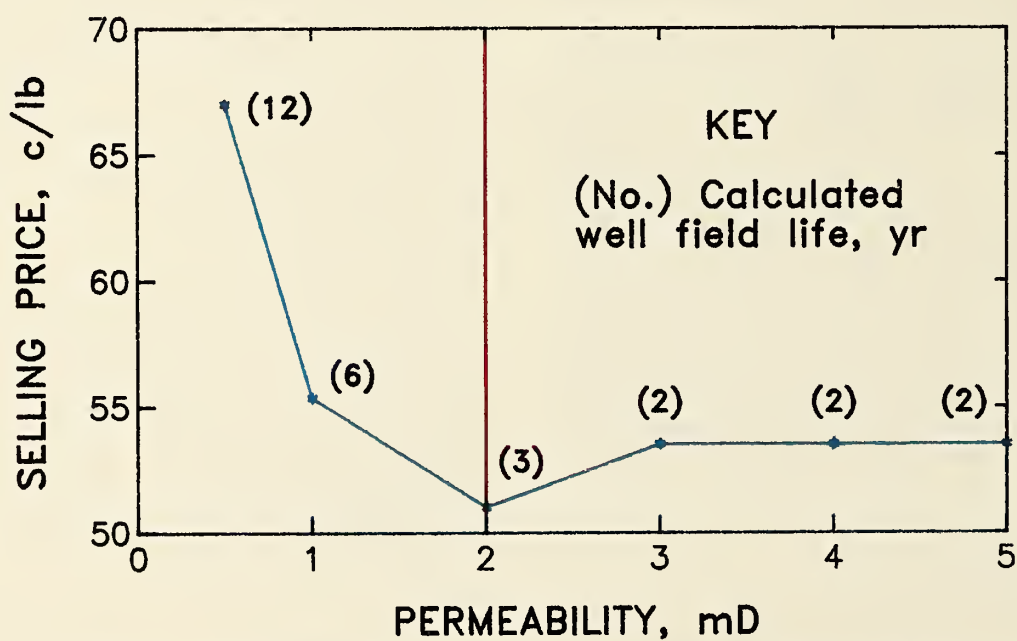


Figure 3.—Combined effects in cash-flow analysis, Santa Cruz commercial-scale operation example. (Vertical line represents initial case.)

SUMMARY

A menu-driven computerized cost model was designed that may be used in developing a commercial in situ copper mining operation. Specifically, this cost model may be used to determine the economics of mining copper resources by the in situ mining method. This model permits the user to make cost inputs and inputs that are converted to mine design elements. Outputs that are generated include specific design parameters and capital and operating costs. Other outputs include a pretax

selling price or DCFROR and a discounted-cash-flow table. The computer model may be used to make sensitivity analyses, whereby one input parameter's value is changed and the other input parameters' values are held constant. The effect of this change on selling price or DCFROR is then determined. The computer program is on an IBM-compatible microcomputer; the minimum hardware requirements for running the program are a 360-Kbyte memory and a math coprocessor.

INTRODUCTION TO THE ENVIRONMENTAL PERMITTING PROCESS FOR IN SITU COPPER MINING

By Daniel J. Millenacker¹

ABSTRACT

This U.S. Bureau of Mines paper introduces the prospective mine operator to the environmental permitting process for an in situ copper mining activity in the State of Arizona. It identifies the environmental aspects of in situ copper mining through a discussion of the regulatory authorization process. The Bureau intends to use this collected information in subsequent phases of its research to obtain permits to conduct a field test at the Santa Cruz, AZ, field site.

INTRODUCTION

In situ copper mining is a relatively new concept, which involves pressure injection of a dilute sulfuric acid leach solution (lixiviant) into an ore zone through a geometrically arranged well pattern. Dissolution of the mineralization located along fractures or disseminated within the ore occurs as the minerals are contacted by the injected lixiviant. Resulting copper-bearing solution is drawn to recovery wells where it is pumped to a surface processing plant. The solvent extraction-electrowinning (SX-EW) method is used to remove copper from solution.

As part of its program to ensure a dependable domestic supply of minerals, the Bureau of Mines is evaluating the use of in situ methods for extracting copper from previously unmined oxide ore deposits. Successful development of this mining technique and its acceptance by the mining industry will require an engineering design both technically feasible and environmentally compatible. One task of the Bureau's research is to identify the environmental issues of in situ copper mining and the corresponding regulatory elements of environmental permitting and protection.

Environmental issues specific to in situ copper mining are addressed through a permitting process administered by Federal, State, and local governmental regulatory agencies. Each agency maintains jurisdiction over a specific environmental element of the mining operation. Permitting involves preparation of comprehensive application documents by the mining company and subsequent document submittal to the appropriate agency. Authorization to proceed with the mining activity may be either granted or denied by the regulatory agency depending upon anticipated facility compliance with environmental protection requirements. Once permits are approved and the facility is constructed, facility operation requires strict adherence to environmental protection performance standards. It is incumbent upon the mine operator to consider all environmental requirements early in

the mine planning process since permitting forms the first major step in establishing the feasibility, costs, and schedule for developing an environmentally sound mine design.

Environmental resources afforded special protection through permitting include ground water, air, biological, and cultural resources. By the very nature of the in situ mining process, the greatest potential for environmental effect rests with the fluid injection-recovery system. Accordingly, the principal environmental permitting issue is protection of ground water resources. A permit application specific to ground water protection must include, at a minimum, information on site characteristics and hydrogeologic conditions, the extent of the area to be affected by the injection activity, anticipated effects to environmental resources, well and surface facility construction specifications and plans for operation, site monitoring methods and procedures, contingency plans, and plans for facility closure. Federal and State requirements restrict injection activities that have the potential to degrade ground water resources. Requirements that apply to ground water protection in Arizona have previously been discussed by Weeks (1).²

Protection of air, cultural, and biological resources are subordinate to the issue of ground water protection. Air emissions expected to result from an in situ operation include fugitive dust from access road traffic, and volatile organics and sulfuric acid mist from operation of the SX-EW facility. Air resources protection requirements are intended to minimize the introduction of contaminants into the open air from a source or operation. Biological and cultural resources may be afforded special protection if ownership of the property upon which the facility is sited is public. Given the potential for effects on ground water and air quality from this type of mining activity, the focus of this paper is on permitting requirements as they apply to protection of these two environmental resources.

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²Italic numbers in parentheses refer to items in the list of references at the end of this paper.

REGULATORY REQUIREMENTS IN ARIZONA

Environmental requirements addressing protection of ground water and air resources in Arizona are currently administered by several governmental regulatory agencies. With respect to ground water protection, the U.S. Environmental Protection Agency (EPA), the Arizona Department of Environmental Quality (DEQ), and the Arizona Department of Water Resources (DWR) maintain jurisdiction. For injection activities occurring on Federal or Indian lands, requirements of the specific Federal land management agency (Forest Service, Bureau of Land Management, or Bureau of Indian Affairs) may also apply. Requirements of these agencies may be addressed in a facility plan of operation or other suitable document. Requirements addressing protection of air resources are administered through either DEQ or county air quality control boards.

GROUND WATER PROTECTION

Federal Water Quality Requirements

EPA regulations relevant to fluid injection activities are authorized under the Underground Injection Control (UIC) program promulgated under part C of the Safe Drinking Water Act, Public Law 93-523. The technical requirements of this program are identified under 40 CFR 144 and 146 (2). Requirements specified under part 144 address permitting and program specifications. Part 146 addresses technical criteria and standards for operation.

According to EPA, no fluid injection shall be authorized if it results in the movement of fluid containing a contaminant into an underground source of drinking water (USDW), if the presence of that contaminant may cause a violation of primary drinking water standards or may adversely affect public health. If an aquifer within a prospective leach interval meets the definition of a USDW, it must be determined to be an "exempted aquifer" before leachant injection may commence. An exempted aquifer is one that would otherwise qualify as a USDW but has been determined to have no real potential as a drinking water source.

Five well classes have been established under the UIC program to regulate fluid injection activities. Two of the five well classes are applicable to in situ mining. A Class III well designation applies to injection wells used for commercial in situ production from ore bodies that have not been conventionally mined. A Class V well is one used for solution mining of conventional mines, such as stope leaching, and for wells used in development of new technologies. The latter well class is generally assigned to those wells that inject nonhazardous fluids and are considered less of an environmental risk. An example of a Class V well is one used during a field-scale test of a mineral deposit to assess viability for commercial-scale in situ production.

For in situ mining, the UIC program authorizes well construction by either permit or rule. Authorization by permit applies to a Class III commercial well facility, while a Class V facility is authorized by rule. Application for a Class III well permit requires explicit and detailed information on the hydrogeologic resources present within the facility area, the well field design and operational plan, reporting and monitoring schedules, and plans for site closure. The principal emphasis of the Class III permit is design and operation of the facility to avoid migration of process fluids into, and degradation of, an

overlying or adjacent USDW. The Class V authorization-by-rule process, on the other hand, requires an operator to provide EPA with generalized inventory information on the facility, along with any additional data that EPA may require. The EPA maintains the prerogative to require a UIC permit for any Class V well authorized by rule at any point in time if operation of that well does not meet minimum operating standards.

In the application for a Class III permit, the physical limits of the study area must be established by defining the "area of review." This establishes the potential area of hydraulic influence of the well field. For a well array, the area of review is defined under 40 CFR 146 as the project area plus a circumscribing area the width of which is the lateral distance from the perimeter of the facility, in which the pressures in the injection zone may cause the migration of the injection and/or formation fluid into a USDW. EPA provides the option of applying either this computational method for determining the area of influence or selecting a fixed radius of not less than one-fourth of a mile around the site.

Closure of a Class III facility requires proper well field abandonment. The need to restore a leached interval is dependent upon the hydrologic conditions expected to return to the site upon completion of leaching activities. If hydraulic conditions are favorable for migration of residual process fluids from the site and the integrity of a receiving USDW cannot be assured, restoration will be necessary. Restoration is the rehabilitation of water quality within the leached interval to a baseline or predetermined water quality standard. A permittee is also required to maintain the financial resources to close, plug, and abandon the well field as required. Financial assurances (performance bonds, financial statements, etc.) must be provided to EPA as a component of the permit application.

Arizona Water Quality Requirements

The State of Arizona presently administers its own water quality protection program under authority of the Arizona Environmental Quality Act (EQA). The EQA was enacted by the State legislature in August 1986 to protect water resources from all activities that have the potential to degrade water quality. The DEQ was established as the principal agency responsible for enforcing provisions of the EQA. Statutory language addressing protection of water resources from injection activities is found under Arizona Revised Statutes (ARS) 49-201 through 208, ARS 49-221 through 225, and ARS 49-241 through 244 (3). The DEQ is required to adopt, by rule, an aquifer protection permit program to control discharges of any pollutant or combination of pollutants that are reaching or may with reasonable probability reach an aquifer. Efforts to establish this permit program are presently under way. With certain exceptions, an aquifer protection permit will be required of any person who owns or operates a facility that discharges fluids. Given the recent date of statutory authorization, the rulemaking process will require 1 to 2 yr before final permit requirements are implemented. The DEQ is also required to adopt, by rule, the UIC permit program described in the Safe Drinking Water Act. Any effort to obtain primacy from EPA for regulating underground injection activities will require the DEQ to adopt rules that closely parallel the EPA-UIC regulations. In developing its own UIC program, a State is not precluded from developing regulations more stringent than those established by EPA. Language specific to State UIC program requirements is identified under 40 CFR 145 (2).

Prior to enactment of the EQA, the Arizona Department of Health Services (ADHS) was responsible for enforcing State water quality protection standards. (The ADHS was the predecessor to DEQ.) The ADHS regulated injection activities through a Notice of Disposal or ground water protection permit. Information requirements for a permit were more comprehensive than those for a Notice of Disposal. Under ADHS, a permit application required information on site hydrogeology and anticipated hydrologic impacts, facility design and operation, monitoring, reporting, contingency planning, and decommissioning. The aquifer protection permit established under the new DEQ regulatory program will require consideration of many of the same elements originally included under the ground water protection permit. A new criterion, however, specifies that an operator apply the best available demonstrated control technology (BADCT) to ensure aquifer protection. BADCT promotes the use of technologies that result in discharge reduction or, if practical, complete elimination of the discharge.

A key element of the new EQA specifies that all water-bearing units in Arizona meeting the definition of an aquifer are to be classified for drinking water protected use. The DEQ has defined an aquifer as a geologic unit that contains sufficient saturated permeable material to yield 5 gal/d of water to a well or spring (4). Water quality standards in effect for an aquifer are the primary maximum contaminant levels established by EPA. The DEQ maintains flexibility to establish aquifer water quality standards for additional pollutants for which no Federal or State standard exists. The classification of an aquifer, or portion thereof, can be changed by DEQ to a use other than drinking water if specific criteria are met. If a leaching operation is to occur in a geologic unit that meets the definition of an aquifer and water quality standards cannot be met at the designated facility point of compliance, the operator must petition DEQ for reclassification of the aquifer (or portion thereof) to a nondrinking water protected use.

Arizona Water Conservation Requirements

A second Arizona agency involved in the permitting process for an in situ copper mining operation is the DWR. The

DWR is principally responsible for ground water conservation but also enforces stringent criteria for drilling, rehabilitation, completion, replacement, and abandonment of wells. Active management areas (AMA) have been established under DWR authority for regulating ground water use. Removal of ground water from a location within an AMA may occur only if a right to that water has been established. The DWR specifies procedures to be followed in establishing a ground water withdrawal right. The DWR criteria specific to water use and well construction are defined under ARS 45-591 through 604 (5) and R12-15-801 through 821 (6).

AIR QUALITY PROTECTION

Construction of an SX-EW plant will require an operator to obtain an air quality installation permit from either DEQ or an appropriate county air quality control board (7). (Arizona counties with approved State air quality control programs are given jurisdiction over facilities that produce a total emission quantity up to 75 st/d. Facilities producing emissions in excess of this rate are regulated by DEQ.) An installation permit is required for each "permit unit" of basic equipment and its attendant air pollution control equipment. A permit unit contains all items of equipment that operate as a single functional unit. Additional permit units may be considered for storage of solids and liquids, incinerators, and fuel burning equipment. The installation permit application must provide a detailed description of all processes, process equipment, storage units, and fuels to be burned, and other information necessary to describe the proposed project and its air pollutant emission sources, together with estimated emission quantities. The permit applicant must describe the system or technique to be used to control emission of all pollutants and additionally provide a description of ambient air quality at the proposed site. An installation permit remains in effect throughout facility construction and facility startup. Once the facility becomes commercially operational, an operating permit is granted by the regulatory agency if specified operational criteria are met. The operating permit establishes the emission standards to be met during facility operation.

IN SITU COPPER MINING FIELD TEST

The Bureau, in cooperation with the Santa Cruz Joint Venture (a mining partnership formed between subsidiaries of ASARCO Incorporated and Freeport-McMoRan Gold Co.), intends to conduct an in situ copper mining field test at the Santa Cruz site in the south-central part of Arizona, approximately 7 miles west of the town of Casa Grande. The project involves construction and operation of two interconnected five-spot well patterns and a surface plant capable of processing copper-bearing leach solution. The facility design was prepared using specifications calculated from the Generic In Situ Copper Mine Design Manual (8). The field test will utilize the unit-cell concept, which is essentially a small well field constructed to commercial-scale well size and spacing specifications. Corner wells in the pattern will serve production needs, and center wells will be used for lixiviant injection. The field test is scheduled to run for a 48-month period. Fluid injection-recovery operations will account for 18 months of this total, while construction, system testing, and decommissioning activities will account for the balance. Although the field test is a scaled-down version of a commercial-size operation, it is sub-

ject to the same environmental analysis as would be required of a full-scale in situ copper mine.

The Bureau, together with the Santa Cruz Joint Venture, has outlined three environmental tasks to be conducted during the 4-yr project life. The first of these is to establish a plan for permitting the field test. This includes developing permitting schedules and identifying the regulatory requirements and design criteria applicable to well field and surface plant construction and operation. The second task is to conduct a site hydrogeologic investigation to define in situ fluid flow conditions and to determine the area of hydraulic influence around the facility. Data collected through this investigation will be used both for permitting and for developing final engineering designs. The third and final task will be to construct the well field and surface plant and operate it in full compliance with permit terms and conditions. This task additionally involves closure of the facility in a manner consistent with permit requirements.

At the present time, the first of the environmental tasks has been completed. Agency jurisdictions have been identified,

and permitting schedules have been estimated. Environmental permits that must be obtained include a Class V well authorization from EPA region 9, an aquifer protection permit from DEQ, a permit to construct wells from DWR, and an air quality installation permit from Pinal County, AZ. The Santa Cruz Joint Venture, as landowner of the property upon which the field test is to be sited, maintains a grandfathered ground water right and is free to withdraw water from the site without obtaining a separate ground water withdrawal permit from DWR. Design and performance criteria of each of these agencies have been incorporated into the preliminary facility design.

A site hydrogeologic investigation, the second task, has been initiated but not yet completed. Collected data will be analyzed and used for development of the DEQ aquifer protection permit, as well as for final facility design. The DEQ permit is by far the most difficult environmental authorization to be obtained for the field test. The estimated time required for permitting is 12 to 16 months. This period is all inclusive from initiation of the site investigation through issuance of the permit. Other agency permits will be sought in a manner consistent with the DEQ permit application schedule. The third task will be initiated during the second year of the field test and will carry through to field test completion.

SUMMARY

The Bureau is evaluating the feasibility of extracting copper from previously unmined oxide ore deposits using in situ methods. This research program gives equal weight to developing a mining method both technically feasible and environmentally compatible.

The Bureau has identified the environmental requirements of in situ copper mining and the regulatory agencies that maintain jurisdiction over fluid injection activities in Arizona. Collected information is being used to develop a program for obtaining all necessary permits to conduct an in situ copper mining field test. The intent of the field test is not only to demonstrate the engineering feasibility of in situ mining but

also to establish the environmental acceptability of this method through permitting, and to operate the facility in full compliance with applicable environmental protection performance standards.

Although the environmental approach to this research is specific to in situ copper mining in Arizona, it can also apply to in situ extraction of other metals or to operations that might occur in other States. The experience gained from this environmental program will provide the mining community with the information necessary to prepare a comparable and effective program for permitting a future in situ mining operation.

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LABORATORY CORE-LEACHING AND PETROLOGIC STUDIES TO EVALUATE OXIDE COPPER ORES FOR IN SITU MINING

By Steven E. Paulson¹ and Harland L. Kuhlman²

ABSTRACT

This paper describes U.S. Bureau of Mines laboratory core-leaching and geologic characterization studies of oxide copper ores. These studies can provide detailed information about the nature of the various chemical and physical processes operating during leaching of oxide copper ores and determine the impact these processes will have on in situ mining. Three types of core-leaching systems are being utilized to conduct these experiments. The leaching experiments provide a very useful means of evaluating the effects of several variables on leaching, such as ore and gangue mineralogy and chemistry, textural relationships between ore and gangue minerals and fluid flow paths, leach solution concentration, flow rate, and pressure and temperature conditions during leaching. Results from two sets of experiments conducted on two different ore types from Pinal County, AZ, are discussed. These ores exhibited very different leaching characteristics because of their distinctly different mineralogy and texture.

INTRODUCTION

The Bureau of Mines has developed a laboratory research program with the main objectives of (1) gaining an understanding of both the physical and chemical factors that have an impact on the in situ mining of oxide copper ores and (2) developing techniques that can be used to evaluate ore deposits for their amenability to in situ mining. Three main research endeavors are utilized in this program: conducting laboratory core-leaching experiments, determining the resulting fluid chemistry from these experiments, and conducting preleach and postleach petrologic studies. The information obtained from such studies on ore from a given deposit is combined with field data from that deposit to provide a better understanding of what can be expected if in situ mining is applied to the deposit.

Up to this point, Bureau core-leaching studies have been conducted only on oxide copper ores from Arizona. The purposes of this paper are to explain in detail how the laboratory core-leaching experiments are conducted so that others may perform similar experiments on other oxide copper ores, to give examples of the types of data that are generated by such studies, and to illustrate the usefulness of these data for evaluating ore deposits for their suitability for in situ mining.

There are two main reasons why the Bureau is conducting leaching experiments on core samples rather than crushed material. First, it is likely that the core-leaching experiments more closely simulate the nature of the solution-rock interface in situ. In many oxide copper deposits, the ore mineralization is distributed in the fractures and pore spaces of the rock, which are where the leach solutions will travel, both in situ and in core-leaching experiments. Thus, the fluid chemistry resulting from such experiments will more

likely resemble that generated during in situ mining, giving a more realistic indication of what the fluid composition will be during in situ mining. In many cases, experiments performed with aggregate material would expose a much greater proportion of gangue mineralization to the leach solutions, perhaps resulting in a leachant chemistry significantly different from that generated during in situ mining. Therefore, it is likely that core-leaching experiments provide a better assessment of parameters such as copper concentration in solution, copper recovery from the rock, and acid consumption by gangue mineralization (1).³

Second, core-leaching experiments allow for an evaluation of permeability changes taking place during leaching. Influences on permeability include mineral dissolution and precipitation, clay mineral hydration, and physical sedimentation. Mineral dissolution may act to increase the size of the flow channels and thus increase permeability. However, the elevated sulfate (SO_4^{-2}) content in the leach solution may result in the combining of SO_4^{-2} ions with cations mobilized by mineral dissolution or ion exchange with clay minerals, causing precipitation of sulfate minerals. This precipitation may decrease permeability. Also, permeability may be adversely affected by any carbon dioxide that may be generated by acid attack on any carbonate minerals in the rock. Although the permeability in a deposit will likely be much different from that measured during laboratory core-leaching experiments, correlation of permeability changes during the experiments with data on the fluid chemistry and sample mineralogy will yield information that may be useful for determining the potential impact of these mechanisms during actual in situ mining in the field.

The three basic requirements for in situ mining are controlled by the inherent petrology and geochemistry of the ore body. First,

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³Italic numbers in parentheses refer to items in the list of references at the end of this paper.

the leach solutions must be able to contact the ore minerals. This is controlled by the rock texture, the nature of the ore mineral distribution, and structural features of the ore body that influence the flow of leach solution through the deposit. Second, the target metals must be selectively solubilized by the leaching reagent. This is accomplished by the type of leach solution utilized and its concentration. Finally, the metallic species must be mobilized to the recovery wells. This is influenced by the solution pH and the concentrations of the various ions in solution, which are controlled by the reactivity of the minerals and the flow rate of the leach solution through the ore body. Precipitation may occur if ion concentration and/or solution pH reaches sufficiently elevated levels. Metal ion mobilization can also be influenced by the interaction of clay minerals with the metal ions in solution, as metal ion sorption onto clay minerals can be significant in reducing metal ion concentra-

tion in solution. Laboratory core-leaching and petrologic studies can help to determine what parameters have the greatest influence on these mechanisms, as well as the magnitudes of these influences. Among the variables being studied are the compositions of the ore and gangue minerals present in the system, the textural and structural relationships among the minerals, the leach solution concentration, the flow rate of the leach solution through the rock (residence time), and the pressure and temperature conditions during the experiment. By incorporating the laboratory data with field data from a deposit, one can make predictions of what will actually occur during full-scale in situ mining of that deposit. Experiments will yield information such as copper loading in solution, fractional copper removal versus time, and lixiviant consumption per unit of copper produced. Such information is essential when assessing the feasibility of applying in situ mining to a deposit.

EXPERIMENTAL APPARATUS

The Bureau has a complete laboratory program to study the petrologic and chemical influences on in situ mining. This program includes the development of sample preparation techniques for both the petrologic studies and core-leaching experiments, as well as several different versions of core-leaching apparatus. Among the desirable features of a core-leaching system are flexibility to accommodate a variety of sample sizes and shapes; nonreactivity of the system with the leach solution; relatively low cost, thereby permitting a large number of samples to be tested; and ability for easy recovery of the postleach sample for petrologic studies (2).

Three systems have been designed to conduct experiments under a variety of conditions. The main difference among the systems is the type of reaction cell used to house the core samples.

Low-Pressure Core-Leaching System With Acrylic Reaction Cell

The first type of reaction cell is constructed from acrylic pipe and can be used at pressures up to 50 psi. It requires a small amount of machining (figs. 1-2) and can be used again for subsequent experiments (2). Portions of the core samples selected for the experiments are submitted for chemical analyses, and thin sections are prepared for examination of the preleach geochemistry and petrology. The remainder of each core specimen is encased in epoxy prior to placement into the reaction cell. This is done because in many instances the samples become very friable because of reaction with the acid, leading to difficulty in preparing postleach thin sections.

The steps in sample preparation are as follows. The sample is first painted with a thin coat of epoxy that cures at room temperature, has low shrinkage, and is not reactive with the sulfuric acid leach solutions used in these experiments (such as 3M 1838-L Epoxy Adhesive).⁴ This thin coating fills all of the surface irregularities of the rock and ensures that the surface is completely sealed. The sample is then placed on end in a segment of polyvinyl chloride (PVC) pipe and the void space filled with epoxy. The inner surface of the pipe is coated with a silicone lubricant prior to the sample insertion to facilitate the removal of the sample after the epoxy has cured. The ends of the cylinder are trimmed to expose the surface of the rock and ground to provide a smooth surface perpendicular to the core axis. (This procedure can also be used for split core samples or other irregularly shaped pieces of ore material if whole core samples are not available.)

After the sample is placed in the reaction cell, O-ring seal caps are placed against each end of the sample. The acid-resistant O-ring

forms a seal between the wall of the reaction cell and the epoxy jacket encasing the sample, which forces the leach solution to travel through the core rather than along the outside edge (i.e., short circuiting). The seal caps contain a hole in the center through which the leach solution is pumped, and the seal cap surface facing the rock is slightly concave to allow access of the leach solution to the

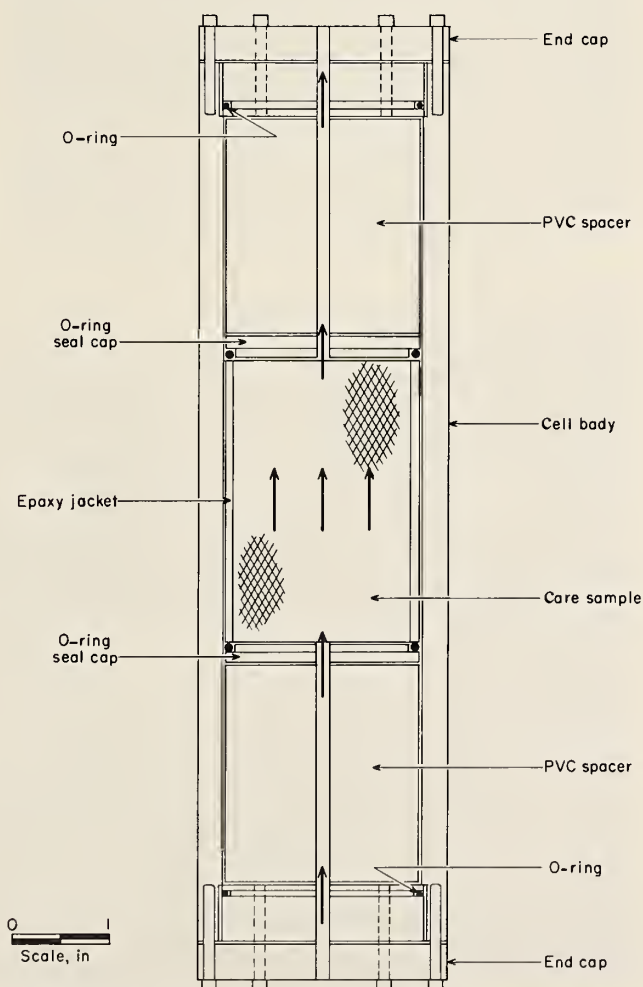


Figure 1.—Cross-sectional diagram of acrylic reaction cell for low-pressure (50 psi) core-leaching experiments.

⁴Reference to specific products does not imply endorsement by the Bureau of Mines.

entire rock face. The necessary force to seal the O-ring at the sample surface is provided by the end caps of the reaction cell. Screws are used to fasten the end caps to the main body of the reaction cell. The end caps are also fitted with O-rings to prevent leakage of leach solution from the reaction cell (fig. 1). If the core sample is shorter than the reaction cell, the excess space is taken up with a solid PVC cylinder of the same diameter, through which a hole has been drilled to permit transport of the leach solution. The solid PVC cylinder transfers the force from the end caps to seal the O-rings at the sample surface. Tubing and fittings made of Teflon

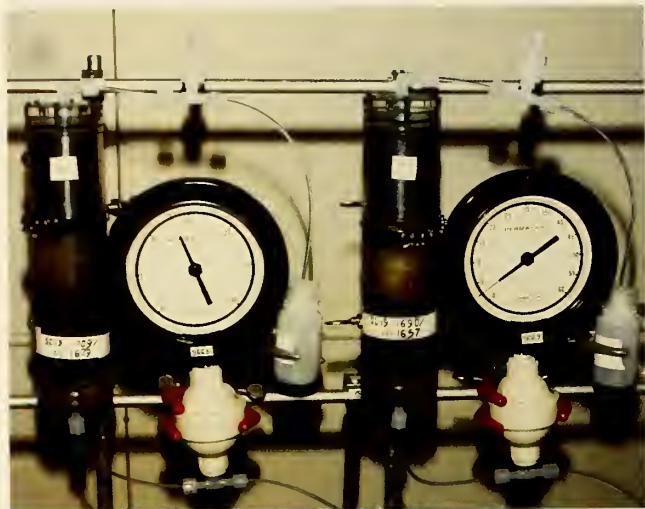


Figure 2.—Two reaction cells for low-pressure core-leaching experiments. Pressure gauges assess permeability changes during the experiment.

fluorocarbon polymer are used for the plumbing on the system, and the fluid samples are collected in closed polyethylene bottles (fig. 2).

The leach solution is injected at the bottom side of the sample, and the injection pressure is monitored with a pressure gauge in order to assess permeability changes that may occur during the experiment (fig. 2). The gauges are fitted with plastic isolators to prevent reaction of the sulfuric acid solutions used in the experiments with the gauges. Reactions between the leach solutions and any components of the leaching system would lead to contamination of the fluid samples and, hence, severely complicate the study of the leachant chemistry and reaction mechanisms operating during leaching. A multichannel peristaltic pump is used to deliver the leach solution to the sample with this system (fig. 3). The pump delivers a constant flow of solution at low pressures (<30 psi), and the flow rate of each channel can be varied by changing the diameter of the flexible tubing that the metal rollers of the pump compress as they rotate.

After completion of the experiment, the core sample is removed from the reaction cell, and the reaction cell can be reused. A portion of the sample is ground and analyzed for its postleach chemical composition. In most cases the sample is quite friable because of degradation during leaching. Therefore, the sample is dried and vacuum impregnated with epoxy so that the cutting and grinding needed for the preparation of the thin section can be easily accomplished. Usually, several thin sections are prepared from each sample to determine the extent of leaching within that sample.

Medium-Pressure Core-Leaching System With PVC Reaction Cell

The second type of reaction cell is constructed from PVC pipe and flanges (fig. 4). The main portion of this reaction cell can be used only once, but it is relatively inexpensive and requires very little machining. When this type of system is used, the sample is

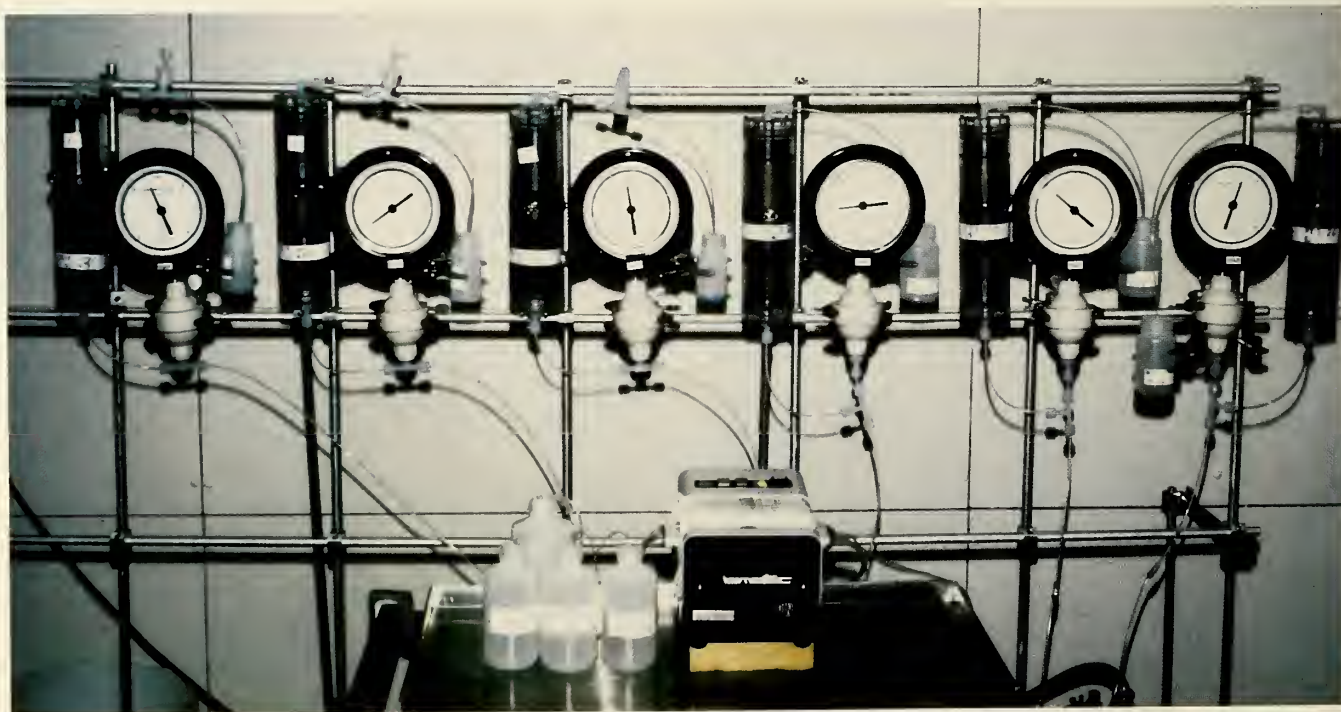


Figure 3.—Low-pressure experimental core-leaching system. This photograph displays six experiments being conducted simultaneously, using a multichannel peristaltic pump to pump the leach solution.

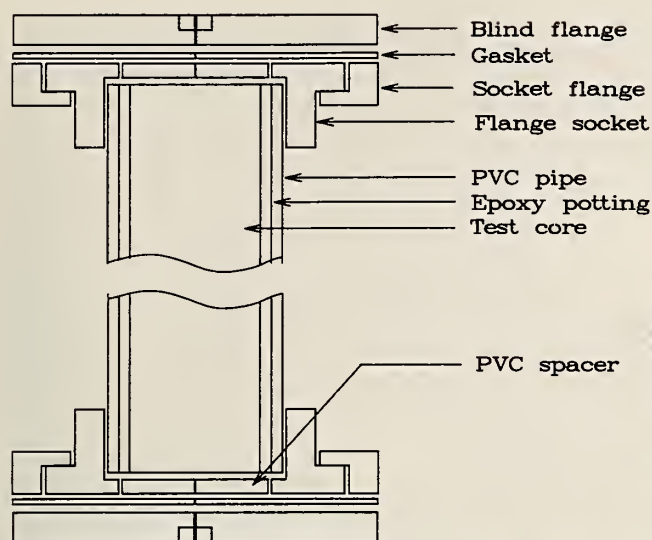


Figure 4.—Cross-sectional diagram of PVC reaction cell for core-leaching experiments.

cast directly into the PVC pipe with epoxy. After the epoxy has cured, the ends of the whole assembly, which consists of core, epoxy, and pipe, are again trimmed and ground to provide two smooth surfaces. Each end is then fitted with a flange assembly, which consists of a flange, a flange socket, a gasket, and a blind flange (fig. 4). The flange itself is attached to the PVC pipe by means of a PVC epoxy adhesive, and the blind flange is attached with steel bolts. In order to reduce the volume of fluid stored in the reaction cell, the void space between the core sample and the blind flange is reduced by using a concave PVC spacer plate, through which a hole has been drilled (fig. 4). Again, as with the other type of reaction cell, a peristaltic pump and Teflon fluorocarbon polymer tubing are used to deliver the leach solution to the core sample when working at pressures below 30 psi.

The PVC reaction cell can be used at pressures up to 150 psi, but it is necessary to utilize a different pump and tubing when working in this pressure range. Since the flow rate of the peristaltic pump is only constant at pressures up to 30 psi, a constant-flow, piston-type pump is used for higher pressure experiments. The pump selected for the current experiments is a constant-flow pump that can deliver fluid at a very low flow rate, if necessary (0.001 mL/min) and can also be used at pressures up to 6,000 psi (fig. 5). This allows for both maximum sensitivity and flexibility. This pump was designed primarily for high-performance liquid chromatography applications.

A pair of piston-type accumulators are used to isolate the pump from the leach solution. Each accumulator has a capacity of 500 mL and is constructed of AISI Type 316 stainless steel (fig. 6). Water is pumped to drive the piston and force leach solution into the core sample. The two accumulators are connected in parallel so that as one delivers leach solution to the sample, the other can be refilled. A series of valves is used to isolate and switch the accumulators. Type 316 stainless steel tubing, fittings, and valves are used in place of Teflon fluorocarbon polymer at pressures greater than 100 psi. All stainless steel wetted parts are passivated with 3M nitric acid for approximately 1 h prior to use in the system in order to further reduce the possibility of reaction with the experimental fluids.

Upon completion of the experiments using the PVC reaction cell, one of two methods is used to remove the sample from the cell: The specimen is overcored using a core bit slightly larger than

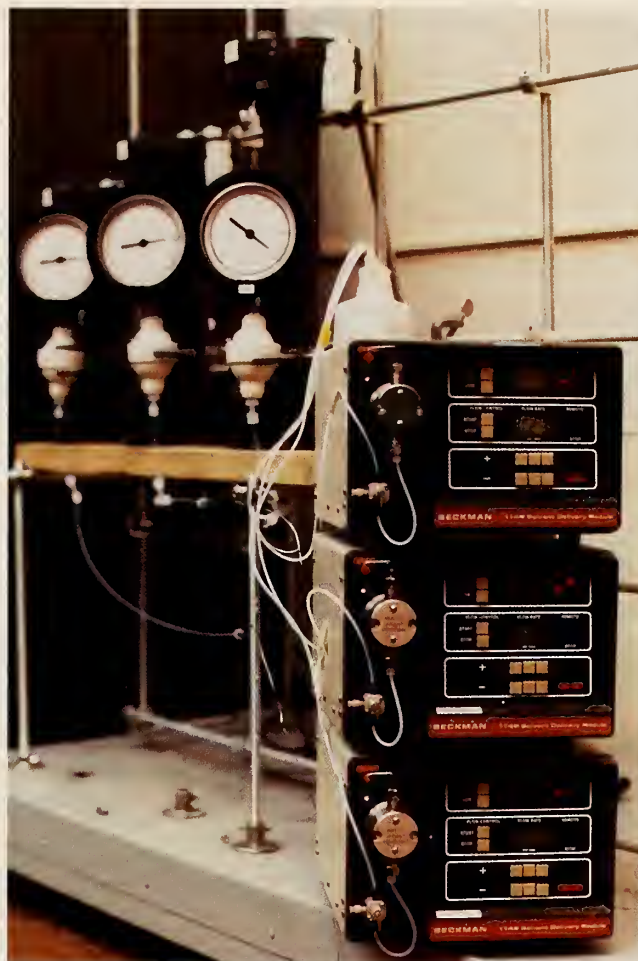


Figure 5.—Three experimental core-leaching systems utilizing PVC reaction cells and high-pressure constant-flow pumps.

the sample diameter, or a lathe is used to trim away the PVC pipe from the sample. After the sample is recovered from the reaction cell, it is prepared in the same manner as described for the low-pressure system, with a sample submitted to the analytical chemistry laboratory before epoxy impregnation and thin section preparation.

Elevated-Pressure Core-Leaching System With Stainless Steel Components

A system has also been developed to conduct experiments at pressures above 150 psi. The maximum pressure at which this system can be used is 1,000 psi. There are two reasons for conducting experiments at these elevated pressures. First, the effects of pressure on the leaching characteristics of various ores need to be assessed, as it is likely that elevated pressures will be encountered during in situ mining of oxide copper ores because of the significant depths of some deposits. Therefore, elevated-pressure experiments would be conducted to determine to what extent, if any, pressure affects such parameters as primary mineral solubility, secondary mineral precipitation, and reaction kinetics. Second, many of the core samples tested in the laboratory exhibit very low permeabilities, thereby making it very difficult or impossible to establish an adequate flow of leach solution through these samples. Thus, a higher pressure is needed to force the leach solutions through the samples than can be attained with either of the two leaching

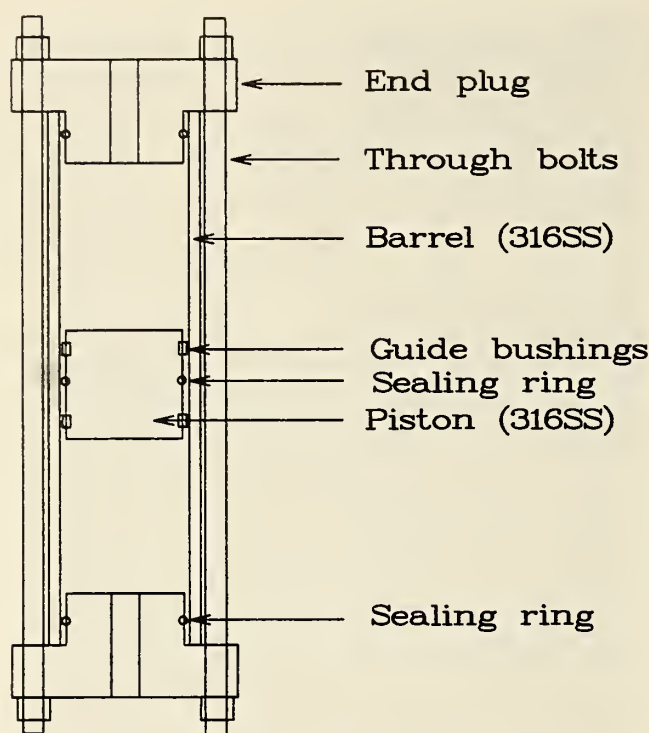


Figure 6.—Cross-sectional diagram of stainless steel accumulator used with high-pressure pump to deliver leach solution to the core sample.

systems previously described. This flexibility to conduct experiments over a relatively wide pressure range allows for a larger variety of samples to be tested and evaluated.

Sample preparation for experiments using the high-pressure reaction cell is similar to that used with the PVC reaction cell. The core sample is placed upright in a piece of ANSI schedule 40 steel pipe, which is then filled with epoxy. After the epoxy has cured, the ends are trimmed and ground to produce smooth surfaces perpendicular to the axis of the sample. End caps for the reaction cell are composed of Type 316 stainless steel and are held in place with threaded steel rod passing through each cap (fig. 7). The end caps are machined to fit over the end of the steel pipe, and they are fitted with an acid-resistant O-ring, which forms the seal between the end cap and the epoxy jacket surrounding the core sample. The end caps are fitted with sintered titanium disks, which act as in-line filters (3) while also providing support to the sample ends, which frequently exhibit some spalling of material from the sample face. Passivated Type 316 stainless steel tubing, valves, and fittings are used throughout this leaching system. The 0.16-cm tubing used has an inside diameter of only 0.06 cm in order to reduce the amount of fluid stored in the system. This small size also results in reduced surface area of the tubing exposed to the leach solution and reduced potential of back-diffusion of metals into the tubing and fluid reservoir. Back-diffusion may occur in experiments with very impermeable samples and, therefore, low flow rates.

As with the PVC reaction cell, a constant-flow, piston-type pump and stainless steel isolators are used to deliver the leach solu-

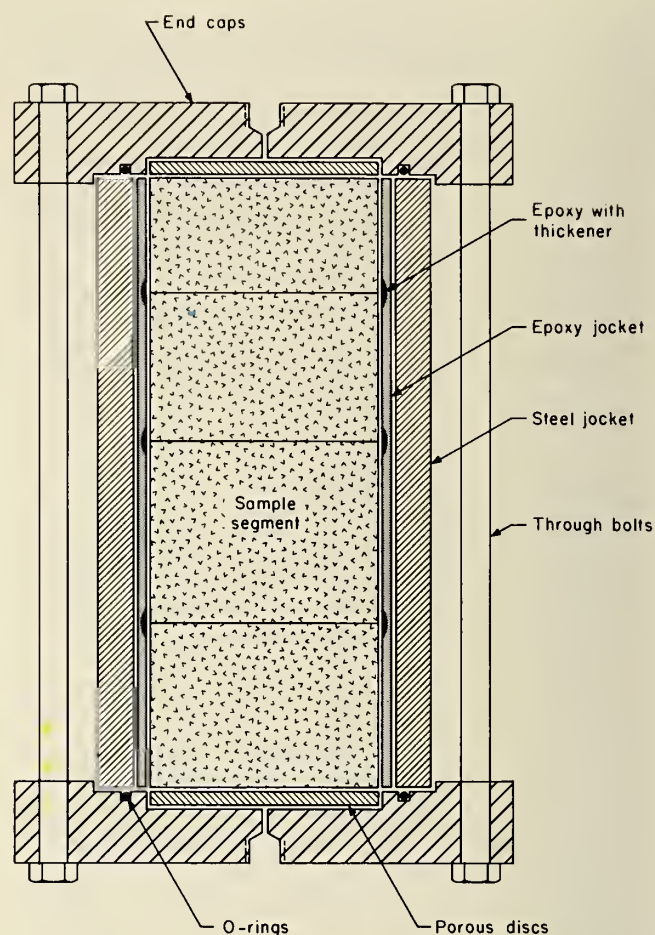


Figure 7.—Cross-sectional diagram of Type 316 stainless steel reaction cell for elevated-pressure core-leaching experiments.

tion to the samples (fig. 8). The pump contains a high-pressure cutoff switch, which is triggered if the pressure reaches the preset upper pressure limit. At this point, the pump then becomes a constant-pressure pump rather than a constant-flow pump, until the pressure again drops below the preset upper pressure limit. The differential pressure between the injection and recovery sides of the sample is measured with a differential pressure transducer that is coupled to an electronic data recorder to provide a continuous readout of pressure throughout the experiment. The injection pressure of the pump is also monitored with the data recorder.

If desired, a back pressure can be applied to the sample by means of compressed nitrogen gas. A pressure gauge is used to measure the actual back pressure applied to the sample. Fluid samples are collected in a Type 316 stainless steel collection vessel, which has been passivated with nitric acid (fig. 8). Experimental fluid samples are removed from the collection vessel on a regular basis and submitted to the analytical chemistry laboratory for analysis. As with the PVC reaction cell, the postleach rock sample can be recovered by overcoring or by turning down the pipe on a lathe.

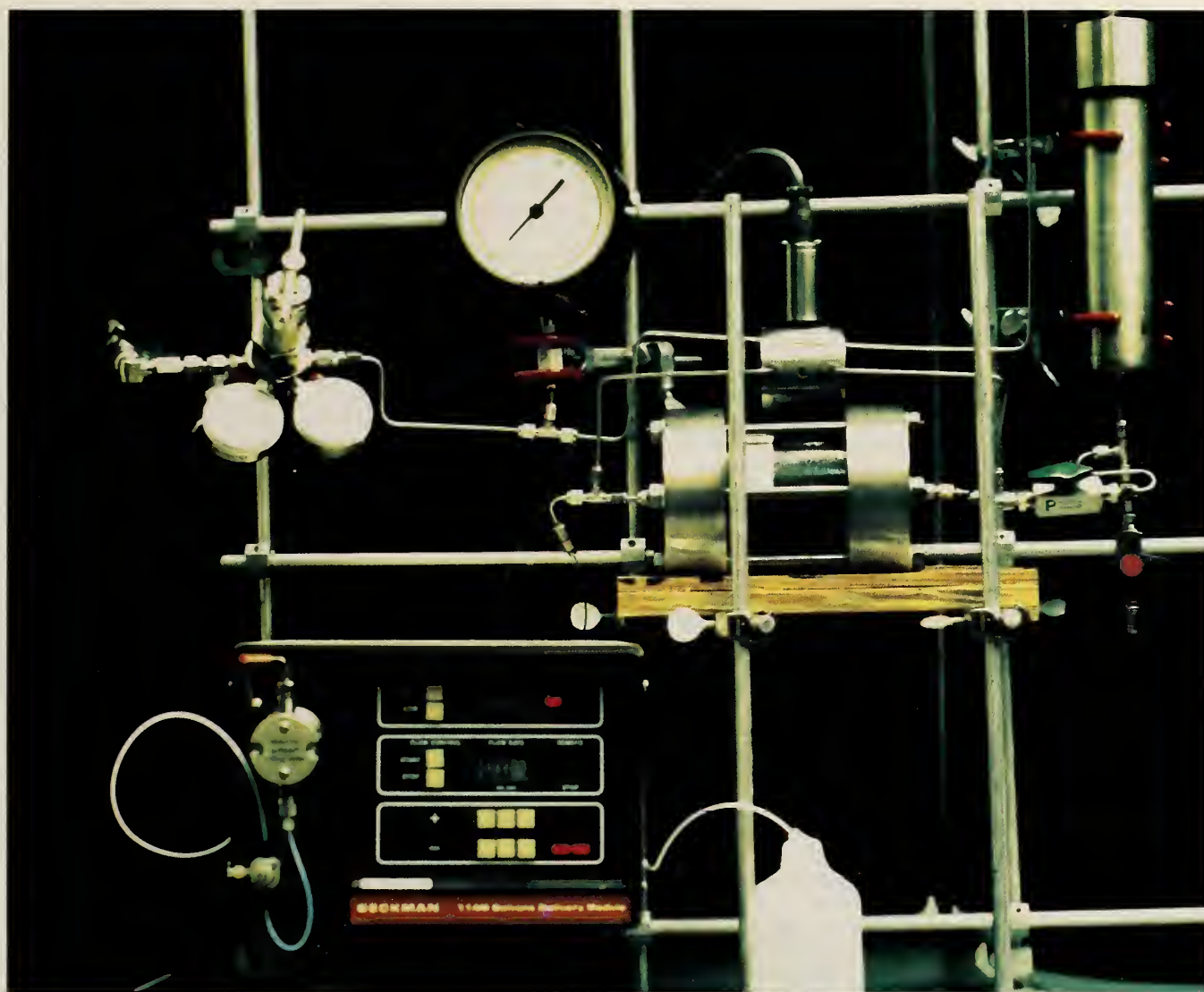


Figure 8.—Elevated-pressure experimental core-leaching system (isolators not shown).

LEACHANT RECIRCULATION EXPERIMENTS

In addition to determining the fluid chemistry from experiments in which fresh leach solution is continuously injected throughout the duration of the experiment, it is also important to determine the effects on leaching chemistry of recirculating the leachants back through the rock after the copper has been removed, since this is what will actually occur during in situ mining. Previous experiments have shown that in addition to copper, appreciable amounts of aluminum, calcium, iron, potassium, and sodium can also be solubilized during leaching of oxide copper ores (1, 4). Also, the amount of SO_4^{-2} in the system increases with time as additional

sulfuric acid (H_2SO_4) is added to the system to replace the acid consumed by gangue mineralization and to maintain the desired pH. Appreciable Cl^- can also accumulate in the leachant if a significant amount of a copper chloride mineral, such as atacamite, is present in the ore. Therefore, experiments were designed to study the effects of fluid recycling on parameters such as mineral solubility, acid consumption, and net permeability.

Leachant recirculation experiments are designed to examine the fluid chemistry after each pore volume of leach solution is pumped through the core. As a convention, pore volume is defined

here as the *initial* pore volume of the sample, since the actual pore volume may change during the experiment because of mineral dissolution and/or precipitation. Approximately 3 to 5 mL of sample is needed for each set of chemical analyses performed on the leachants. Therefore, it is desirable to recycle a total volume of solution that will not result in significant depletion of fluid because of sampling for the analytical determinations. Thus, in order to ensure that an adequate amount of fluid is available for the duration of the experiment, a core sample with a pore volume ≥ 100 mL is suggested. Since the porosity of many of the oxide copper ores tested in the laboratory ranged from 5 to 10 pct, a total rock volume between 1,000 and 2,000 cm³ is recommended. Therefore, two types of experiments have been designed: one that utilizes a large-diameter core sample (e.g., 10 or 15 cm) and one that uses a longer segment of standard 5-cm-diameter core.

A large-diameter core sample is more desirable than a smaller diameter sample because it has a greater surface area over which to inject the leach solution, thus enhancing the possibility of intersecting mineralized fractures and fluid flow channels. Use of a large-diameter sample also decreases the length of core necessary to meet the minimum sample volume requirement, thereby allowing for a faster flow rate than in a smaller diameter core sample with the same permeability. This faster flow rate results in a reduction in the time necessary to complete an experiment. Since 10- or 15-cm core is not commonly available from exploratory drilling, larger blocks of ore obtained from such sources as open pit or underground mining operations need to be drilled to obtain these large-diameter core samples. However, it is possible to conduct such recycle experiments using smaller diameter core if larger samples are not available.

Either the PVC or elevated-pressure reaction cell can be used for the recycle experiments. The same basic design is utilized, although the sizes of the components in the system need to be matched with the diameter of the core samples to be used in the experiments. After the diameter and approximate porosity of the core samples have been determined, the length of core needed to meet the minimum volume requirement can be calculated. In many cases it will be necessary to use several pieces of core in one reaction cell in order to meet the minimum sample size required to conduct the recycle experiment. The ends of core samples selected for the experiment are trimmed on a rock saw and ground to provide flat and reasonably smooth surfaces. The samples are then stacked upright in the desired order, and a continuous composite rock core

is created by using epoxy adhesive, thickened with amorphous fumed silica (3 wt pct), to spread around the joints between adjacent core segments. The fumed silica, which is unreactive with the acidic leach solutions, gives the epoxy a pastelike consistency, which allows it to be spread across the joints between sample segments (fig. 7). Care must be taken to ensure that the thickened epoxy is not forced into the interface between the sample segments, as this will result in reduced solution flow through the composite sample. After this thickened epoxy has cured, the composite core sample is cast in epoxy and prepared as previously described.

After an adequate amount of leachant has been collected and a 3- to 5-mL sample has been removed for the analytical chemistry determinations, one of two methods is used to remove the copper from the leachant prior to reinjection. The first method utilizes a small electroplating cell to plate the copper directly out of the leachant. The leachant is placed in a beaker and stirred as copper is plated onto a copper cathode. A platinum basket is used as the anode in this system. The second method of copper removal utilizes an organic solvent extractant. Examples of solvent extractants that worked well are Henkel LIX-622 and Acorga M-5397. A mixture consisting of two parts of kerosene and one part of extractant is added to an equal volume of leachant in a separatory funnel. The mixture is agitated for several minutes and then allowed to settle before the leachant is drawn off. This procedure is repeated until the leachant no longer exhibits any blue color (usually 2 to 5 times, depending on copper concentration). The leachant is then filtered through a 0.45- μ m membrane filter to remove additional organic material that may be suspended.

The process of removing copper from the leachant results in an exchange of H^+ for Cu^{+2} . However, the H^+ consumed by gangue mineralization is not returned to solution during the copper removal step. Therefore, it is necessary to restore the acid concentration to its initial value. This can be achieved by adding either a relatively small amount of concentrated sulfuric acid or a larger amount of dilute sulfuric acid. Which option to use will be dictated somewhat by the total volume of solution being recycled. Adding concentrated sulfuric acid can cause a rapid increase in the SO_4^{-2} concentration, which may be undesirable. However, the addition of dilute sulfuric acid may lead to undesirable dilution of the leachant. The advantages and disadvantages of both methods should be considered. Perhaps the most important point is to maintain consistency among experiments, as this will simplify data interpretation and comparison.

CHEMICAL AND PHYSICAL MEASUREMENTS

Prior to leach solution injection, distilled, deionized water is pumped through the sample to completely saturate it and establish the flow rate conditions for the experiment. Fluid samples from the leaching experiments are collected, weighed, and analyzed on a regular basis, generally every 48 to 72 h, to thoroughly document the exact flow rate and chemical composition throughout the duration of the experiments. This sampling frequency was chosen to ensure that a large number of data points are obtained for each experiment. The acidity of each sample is measured using two different methods. The first is a standard pH measurement and the second is an end-point titration technique to measure the free-acid content of the samples. Free acid is defined here as the H^+ present in the leachant, excluding any contribution due to cation hydrolysis after leaching (4). Because of the elevated levels of the various cations in solution, the potential exists for appreciable H^+ generation by hydrolysis reactions with these cations after the leachant is collected from the experiment. Such reactions may lead to erroneous assessments of acid consumption during leaching.

Therefore, a potassium oxalate buffer solution is added to a small subsample of the leachant to complex the cations present. This solution is then titrated with a base to determine the actual amount of H^+ remaining in the leachant collected, and therefore, how much of the initial acid was consumed by reaction with the rock. An automated titration system is utilized, which enables the rapid titration of many samples.

Approximately 2 to 3 g of leachant is diluted with distilled, deionized water, acidified, if necessary, to prevent any precipitation after dilution, and analyzed for the following components: dissolved silica, and aluminum, calcium, copper, iron, magnesium, potassium, and sodium. Atomic absorption (AA) spectroscopy and inductively coupled argon plasma (ICAP) spectroscopy are used to perform the analytical determinations for these components. An additional 0.25 to 0.50 g of solution is diluted and analyzed for SO_4^{-2} and Cl^- using ion chromatography. Past experience has shown that the leachant chemistry from the experiments using oxide copper ores is composed chiefly of these 10 components. However,

the specific analytes to be measured should be determined after a close examination of the mineralogy of the rock samples to be used for the experiments. Also, when conducting recycle experiments, it may be necessary to expand the number of analytes, as elements present only in trace amounts in the rock may accumulate to appreciable concentrations after several recycle steps.

Prior to the initiation of each experiment, a portion of the core sample is crushed, ground to pass a 100-mesh sieve, decomposed by a fusion or acid dissolution technique, and then analyzed for its elemental composition. ICAP and AA spectroscopy are used to measure aluminum, calcium, copper, iron, magnesium, manganese, potassium, silicon, sodium, and titanium. Combustion techniques are used to measure carbon and sulfur, while ion chromatography is used to measure chlorine in the rock. Trace elements such as antimony, arsenic, barium, cadmium, chromium, lead, mercury, selenium, silver, and zinc are measured using graphite furnace AA spectroscopy.

Preleach and postleach petrologic and microprobe examinations of the ore samples provide a great deal of useful information about the texture and chemistry of the samples, which can be combined with the results of the leaching experiments to help explain the reactions occurring during leaching. Polished thin sections of the preleach and postleach samples are prepared and examined using a polarizing microscope. Qualitative and quantitative chemical information is obtained using an electron probe microanalyzer equipped with an X-ray analyzer. This system has the capability to analyze an area only a few micrometers in diameter, as well as X-ray mapping capabilities, which are used to display elemental distributions within a sample.

EXPERIMENTAL RESULTS AND DISCUSSION

Fluid chemistry data generated in the core-leaching experiments are evaluated to provide input for making decisions in the field on such parameters as the number of wells operating at any given time, well spacing, and residence time of the fluid in the ore body. These three parameters have an effect on the copper loading in solution and the recovery rate of copper from the deposit. The minimum copper concentration in solution necessary to ensure the efficient operation of the surface recovery facility is maintained by controlling these three parameters. Among the mechanisms that influence the copper recovery rate are (1) chemical kinetics of mineral-acid reactivity, (2) acid and copper diffusion through reaction product layers, (3) acid and copper diffusion between the main flow channels and the other pores and microfractures in the rock, (4) chemical equilibria affecting the mobilization of copper to the recovery wells after dissolution, particularly pH controls, and (5) adsorption of mobilized copper onto mineral surfaces in the rock [Davidson (6)]. Core-leaching experiments provide a measure of the net effect of these mechanisms, which can then be used as input for the design of a true in situ mining well field.

Some examples of data from both the core-leaching experiments and the petrologic studies are presented here to illustrate the type of data such studies yield and how this information is evaluated with respect to in situ mining. Each ore deposit displays its own unique characteristics; however, if the fundamental reaction mechanisms operating during leaching can be identified and quantified in the laboratory, then the evaluation of a particular deposit for in situ mining will be simplified. Thorough petrologic and electron microprobe studies of the rock samples both before and after the experiments provide information that is very useful in explaining the resulting leachant chemistry and identifying the various reactions operating during leaching.

In addition to the chemical analyses, porosity and permeability measurements are performed on the core samples used in the laboratory experiments. To determine the effective porosity of a sample, it is first dried under vacuum and weighed, followed by a vacuum saturation with distilled, deionized water and a second weighing. After the total volume of the sample is measured, the effective porosity is calculated using the volume of pore water after saturation. The permeability of each sample is determined after the injection of distilled, deionized water is initiated at the beginning of the experiment. The following equation, derived from Darcy's law, is used to calculate the permeability (5):

$$K = \frac{nVL}{APt}$$

where K = permeability, D,
 n = viscosity of fluid at measured temperature, cP,
 V = volume of fluid, cm^3 ,
 L = length of sample, cm,
 A = cross-sectional area of sample, cm^2 ,
 P = pressure, atm,
 and t = time, s.

The field permeability of a particular ore deposit will most surely be quite different from that measured in a core sample in the laboratory, owing to the appreciable impact of large fractures and faults occurring in the deposit. However, the main purpose of measuring and monitoring permeability during the experiments is to assess the *relative* changes in permeability that occur as the experiment progresses, rather than to focus on the absolute permeability of the core samples.

To illustrate the significance of mineralogy and texture on the leaching characteristics of an ore, results of experiments using two oxide copper ores with distinct differences in their texture and mineralogy are presented, which show the very different reactivities that these ores exhibit. Both ores were from deposits in Pinal County, AZ.

Chrysocolla Hosted in Granodiorite Porphyry

The first type of ore studied was a granodiorite porphyry that contained chrysocolla as the principal copper mineral. The porphyry is significantly altered, resulting in a permeable, porous rock. Thorough petrologic studies of many thin sections of preleach samples determined that the average mineralogy of this ore was 33 pct quartz, 30 pct plagioclase feldspar (one-half altered to copper-bearing clay), 16 pct potassium-feldspar, 12 pct biotite (one-half altered to chlorite and clay), 6 pct chrysocolla in fractures, and 3 pct limonite. Chemical analyses of this ore resulted in an average copper content of 2.3 wt pct. To assess the potential copper removal, it is important to determine the extent and distribution of the copper in various minerals. This particular ore displayed the following copper distribution: 55 pct in chrysocolla, distributed mainly in fractures but also in altered plagioclase feldspar phenocrysts; 40 pct in clay minerals, distributed in altered plagioclase feldspar and biotite phenocrysts; and 5 pct in limonite mineralization (1) (fig. 9). Copper mobilization from these various minerals is dependent on both the accessibility of the ore minerals to the leach solution and the mineral solubility. The textural relationships between the copper-bearing minerals and the fractures in the rock will determine the leach solution accessibility. Copper mineralization located along fractures (figs. 9-10) will have greater accessibility to the

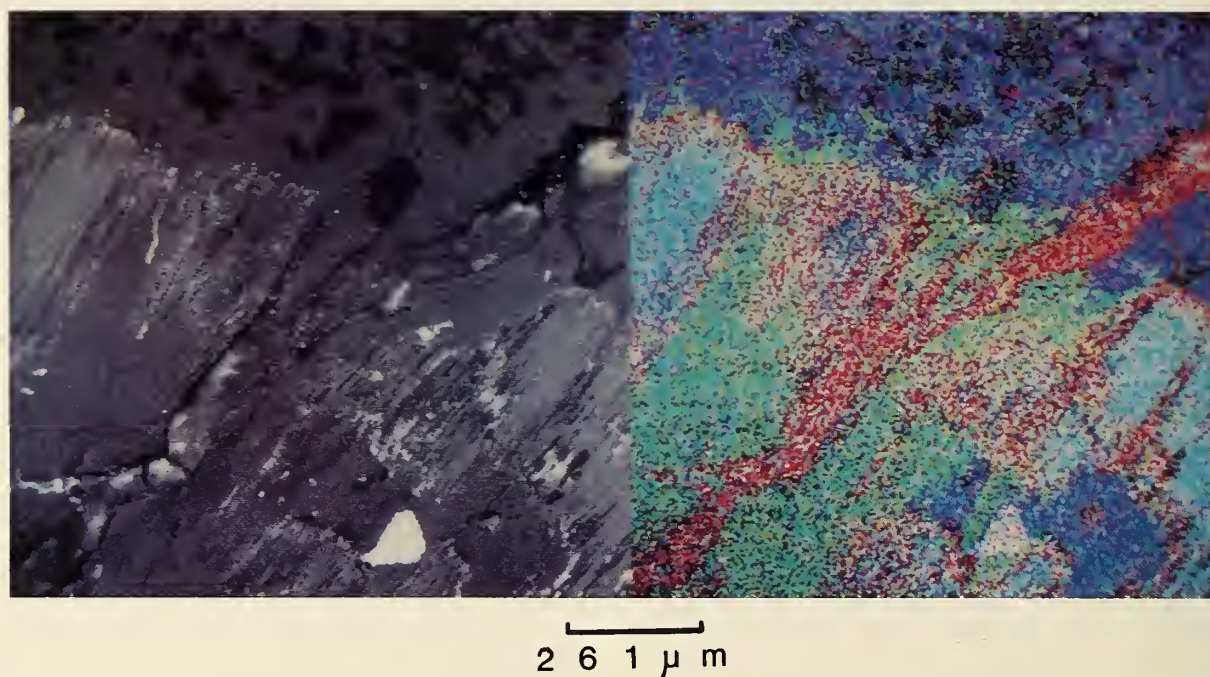


Figure 9.—Left, backscattered electron image (BEI) of chrysocolla veinlet (center) crosscutting an altered biotite phenocryst in preleach sample of granodiorite porphyry ore. Right, BEI image with computer-enhanced X-ray overlay maps that show the distribution of single elements or combinations of elements: copper (red), iron (green), potassium (dark blue), and iron and potassium (light blue). Copper is distributed in this sample not only in chrysocolla but also in the altered biotite. The dark-blue area at the top of this map is potassium-feldspar. (These images were obtained from polished thin sections on the electron probe microanalyzer. The colorized X-ray dot maps were generated by rastering an electron beam across the sample and counting the X-ray emissions of the selected elements with an energy-dispersive spectrometer.)

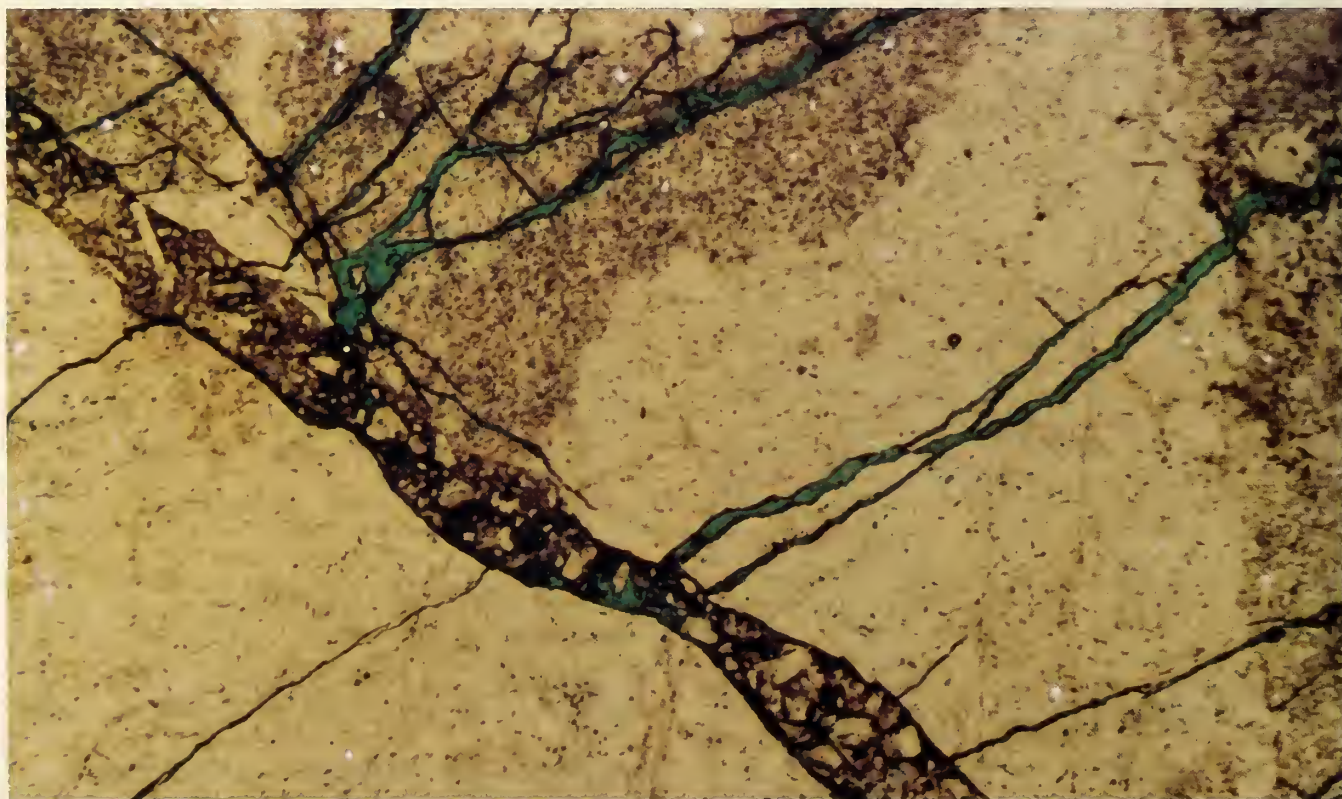


Figure 10.—Quartz monzonite crosscut by numerous fractures filled by atacamite and clay mineralization; photomicrograph.

leach solution than will disseminated copper mineralization (fig. 11), especially if in the latter case the matrix permeability and fracture density are relatively low. Postleach petrologic examinations of areas that were contacted by the leach solution will also yield information on the relative solubilities of both the ore and gangue minerals.

Three experiments were conducted on the chrysocolla ore using three different leach solution concentrations: 5, 15, and 25 g/L sulfuric acid (1). These experiments were conducted in the acrylic reaction cells with the leach solution delivered to the samples with a peristaltic pump (low-pressure system described earlier). Leachant was not recirculated during these experiments; thus, fresh leach solutions were continuously injected into the samples. The core samples were approximately 13 cm in length and had diameters near 5 cm. The experiments were conducted at ambient temperature and pressure conditions.

Figures 12, 13, and 14 display the resulting fluid chemistry from experiments conducted on this ore at the three different concentrations of sulfuric acid. When this information is combined with the results of the preleach and postleach petrologic and microprobe studies, conclusions can be drawn about the various reactions operating during leaching. For example, the graph of copper concentration versus time (fig. 12) reveals that no copper was mobilized during the first 200 to 500 h of the experiment. However, appreciable amounts of calcium and sodium were mobilized during these initial stages of the experiments (figs. 13-14). X-ray diffraction and microprobe studies of the clay minerals in this ore identified significant calcium and sodium in the exchangeable sites of the clays. This, along with the elevated pH of the fluids during this

time, led to the conclusion that no copper was mobilized in the initial period of the experiments because the acid was totally consumed by ion-exchange reactions taking place, which left no H^+ available for copper dissolution. After H^+ replaced all the exchangeable cations in the clays, as evidenced by the rapid decrease in calcium and sodium in solution, copper dissolution proceeded. Thus, if a significant amount of clay mineralization is present in a deposit, there exists a potential for considerable acid consumption. The fact that there is an initial delay in copper production, yet high reagent consumption, may have a significant impact on the economic analysis of a deposit. Therefore, initial microprobe and leaching studies will help to quantify the significance of ion-exchange reactions for a particular ore. The Bureau is currently investigating various solutions that may be injected into the ore prior to acid injection to reduce or eliminate the impact of ion-exchange reactions on the leaching process.

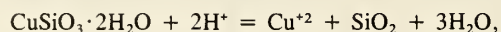
The leachant chemistry (figs. 13-14) and postleach petrologic analyses indicate that the biotite in this rock undergoes attack by the sulfuric acid leach solutions and contributes aluminum, iron, magnesium, and potassium to the leach solutions. While some aluminum and iron were removed from the clay minerals and limonite, respectively, biotite dissolution and the initial ion-exchange reactions in the clay minerals were the main mechanisms occurring in this rock that consumed acid. However, even though biotite dissolution was occurring, postleach microprobe studies revealed that only a minor amount of the sorbed copper present in the biotite (fig. 9) was removed. Most of the copper in solution was contributed by chrysocolla and altered plagioclase phenocrysts, with a minor amount from limonite (7). The implications of this are significant,



Figure 11.—Quartz monzonite containing disseminated atacamite mineralization; photomicrograph.

as previous studies have found that as much as 15 pct of the total copper in a sample is present in the biotite (7). Also, most of the copper contained in the clay minerals was solubilized during leaching, with little evidence of significant sorbed copper remaining in the clay minerals examined in the postleach samples. This again illustrates the value of the information obtained from the leaching and petrologic studies.

In order to obtain the required information to perform an economic evaluation of in situ mining on a larger scale, it is necessary to determine the consumption of acid per unit of copper produced and the dependence of acid consumption on leach solution concentration. For example, using the general equation for chrysocolla dissolution by sulfuric acid:



the stoichiometric acid consumption is calculated to be 1.54 g sulfuric acid per 1 g of copper removed. By comparing the acid consumption observed in each experiment with the calculated stoichiometric value, one can evaluate the severity of gangue mineral acid consumption and draw correlations between consumption and leach solution concentration. As previously discussed, experiments performed on the granodiorite porphyry ore resulted in an initial very high consumption of acid due to ion-exchange reactions with the clay minerals (fig. 15), which decreased rapidly as the ion-

exchange reactions neared completion. As copper mineralization becomes depleted, gangue mineral consumption of acid steadily increases, and it is proportional to acid concentration. Graphs such as figure 15 illustrate that core-leaching experiments can help to evaluate the reactivity of various types of ores and can also aid in determining the optimum leach solution concentration for specific ore types, as acid consumption will have to be weighed against copper loading in solution and copper recovery rate (1).

The net permeability of the three samples increased during all three experiments, despite the fact that sulfate precipitation occurred. Sulfate concentrations in the leach solutions revealed that sulfate precipitation was occurring early in the experiments, before the concentrations rebounded to the initial level of the fresh leach solution. These depletions corresponded to the initial elevated calcium concentrations observed in the experiments. Therefore, it is likely that gypsum precipitation occurred during this time, as gypsum was observed in the postleach core samples. Also, the first few leachant samples contained gypsum that had precipitated from solution. However, the dissolution of chrysocolla had a stronger influence on permeability, resulting in the net increases in permeability for the three experiments. These increases ranged from 4 to 110 times the initial value, with the large increases resulting from the observed dissolution of fracture-hosted mineralization. It should also be noted that no chloride was present in the leachant because of the absence of chloride mineralization in this ore type.

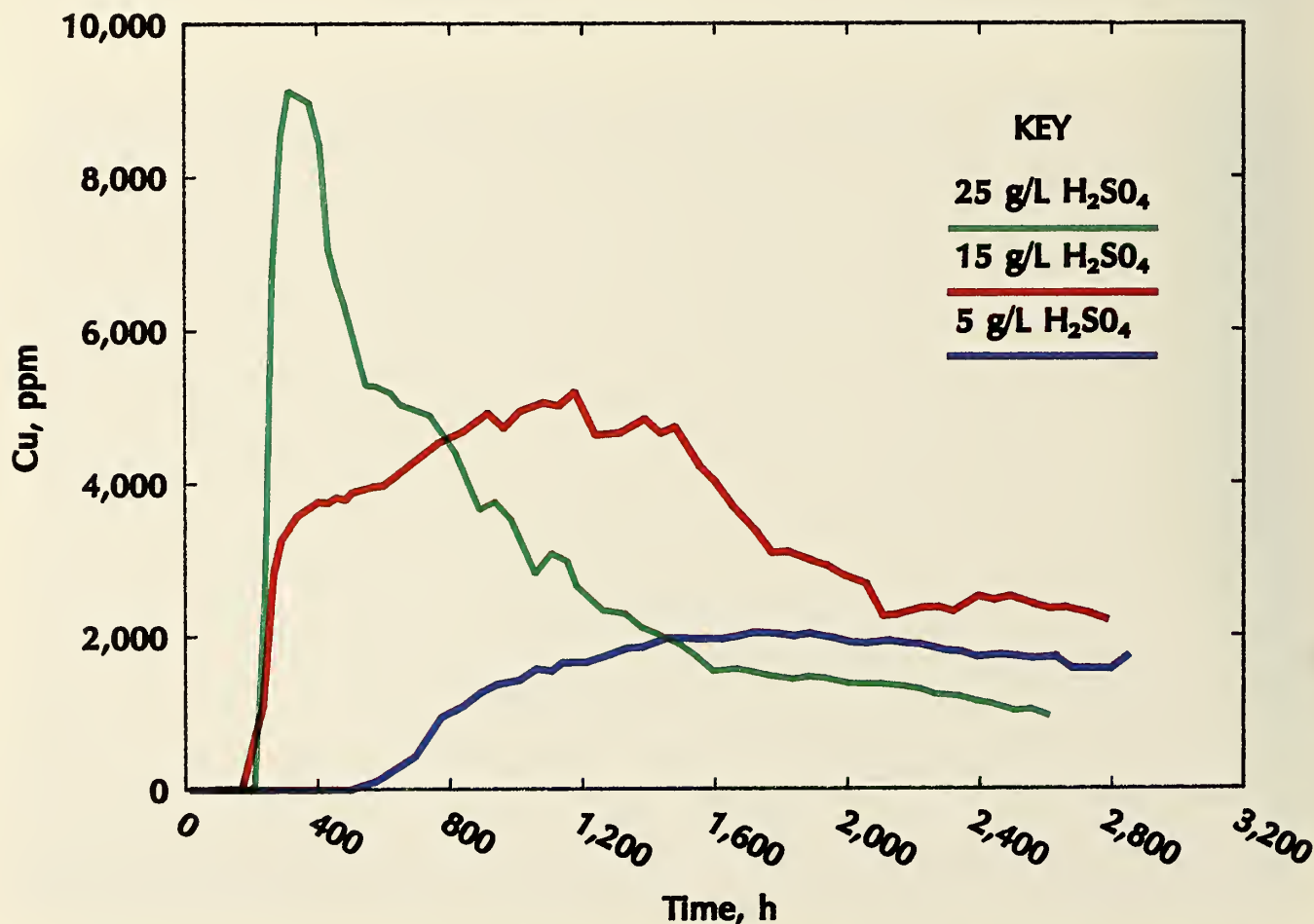


Figure 12.—Copper concentration in solution versus time for three core-leaching experiments using granodiorite porphyry-hosted chrysocolla ore.

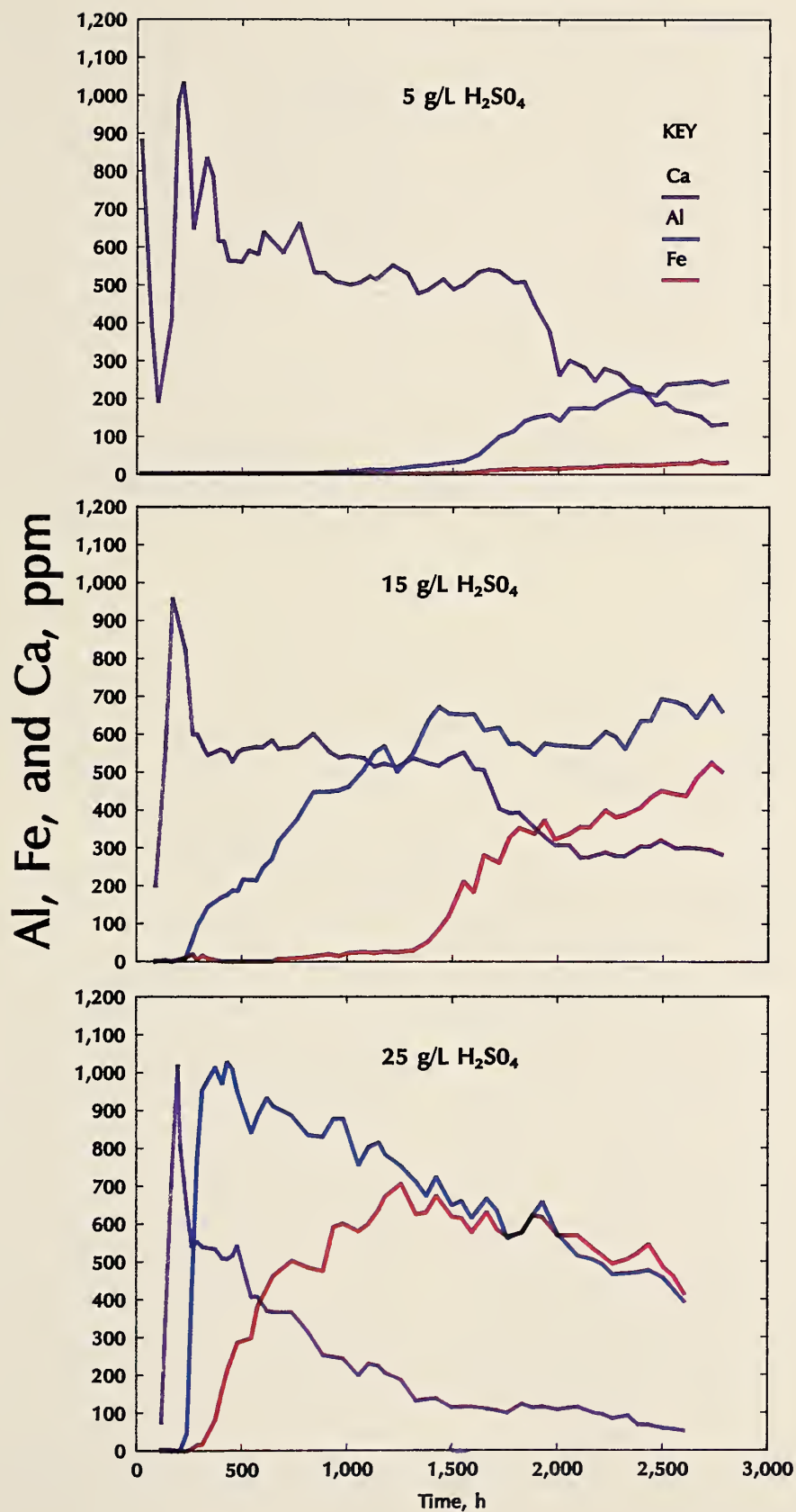


Figure 13.—Aluminum, calcium, and iron concentration in solution versus time for three core-leaching experiments using granodiorite porphyry-hosted chrysocolla ore.

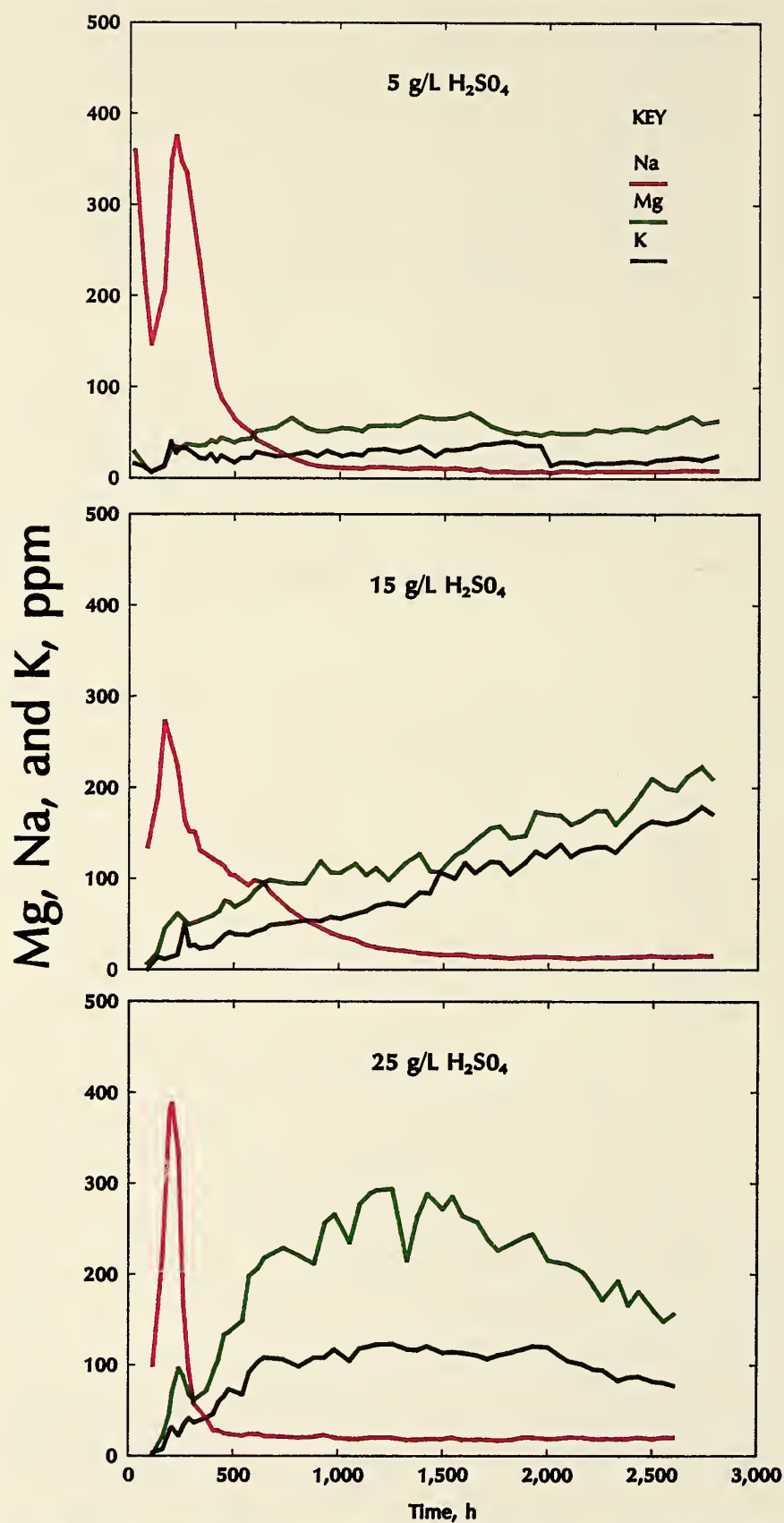


Figure 14.—Magnesium, potassium, and sodium concentration in solution versus time for three core-leaching experiments using granodiorite porphyry-hosted chrysocolla ore.

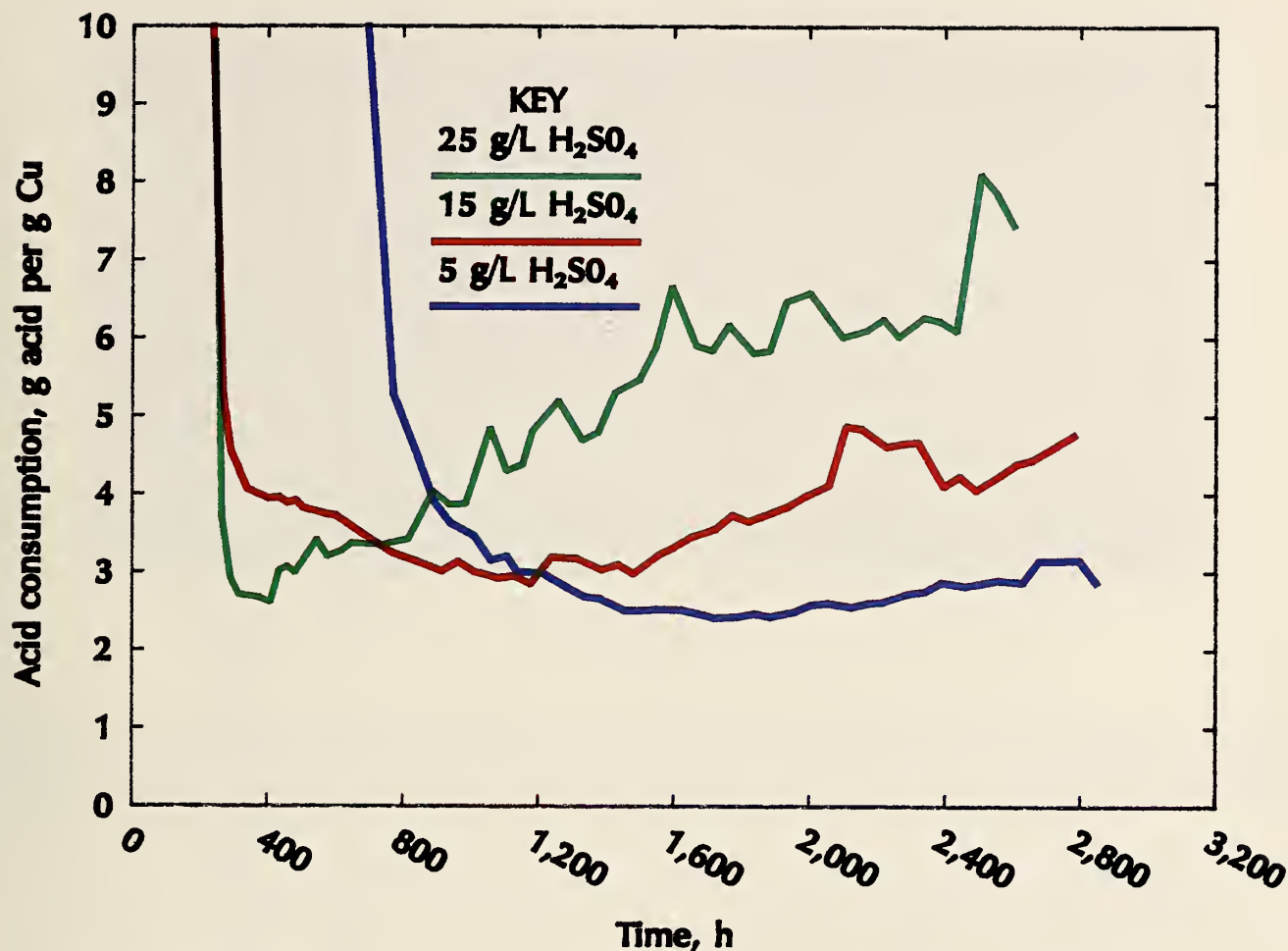


Figure 15.—Acid consumption versus time for three core-leaching experiments using granodiorite porphyry-hosted chrysocolla ore.

Atacamite Hosted in Quartz Monzonite

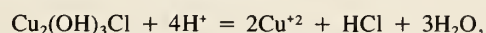
A second type of ore examined was a quartz monzonite, which contained atacamite as the predominant copper mineral. The rock is medium to coarse grained and exhibits a relatively low matrix permeability. Petrologic studies of preleach thin sections determined that the average modal mineralogy of this ore was 40 pct quartz, 25 pct potassium-feldspar, 20 pct sericite, 5 pct atacamite, 5 pct limonite, and 5 pct clay mineralization. The copper content of these samples ranged from 3 to 9 wt pct, with the large variability attributable to the substantial fracture density variations from sample to sample. Unlike the granodiorite porphyry-hosted chrysocolla ore previously discussed, nearly all of the copper in this rock is located in fracture-hosted atacamite mineralization (figs. 10, 16), with minor amounts distributed in the clay minerals and limonite (1). These differences in mineralogy and texture result in very different leaching characteristics from those displayed in the chrysocolla ore.

Three experiments were conducted on this ore using three different leach solution concentrations: 10, 20, and 40 g/L sulfuric acid (1). These experiments were also conducted in the acrylic reaction cells, with the leach solution delivered to the samples with a peristaltic pump. Leachant was not recirculated during these experiments; thus, fresh leach solutions were continuously injected into the samples. The samples used in these experiments were split pieces of 5-cm-diameter core approximately 6 cm long. These

experiments were also conducted at ambient temperature and pressure conditions.

Because of the high ore grade, the distribution of nearly all ore mineralization in fractures, and the relative absence of acid-consuming gangue minerals, copper loadings in solution were very high in these experiments (fig. 17). The absence of acid-consuming gangue mineralization is indicated by the fluid chemistry from the experiments (figs. 18-19) and was also supported by petrologic studies. Only a small number of ion-exchange reactions with clays occurred, as shown by the rapid decrease in calcium, potassium, and sodium in solution. A relatively small amount of iron was mobilized as the result of limonite dissolution. The absence of biotite in these samples resulted in virtually no magnesium mobilization.

The low reactivity of the gangue minerals in these samples is reflected in the graph of unit acid consumption versus time for the experiments (fig. 20). The average consumption for the three experiments ranged from 1.2 to 1.5 g sulfuric acid per gram of copper. In the dissolution of atacamite,



it is seen that hydrochloric acid is generated. Thus, 1.5 mol of sulfuric acid (3H^+) is consumed in the production of 2 mol of copper, resulting in an acid consumption of 1.16 g sulfuric acid per 1 g of copper removed. Thus, the acid consumption observed in these experiments is very close to this stoichiometric consumption.

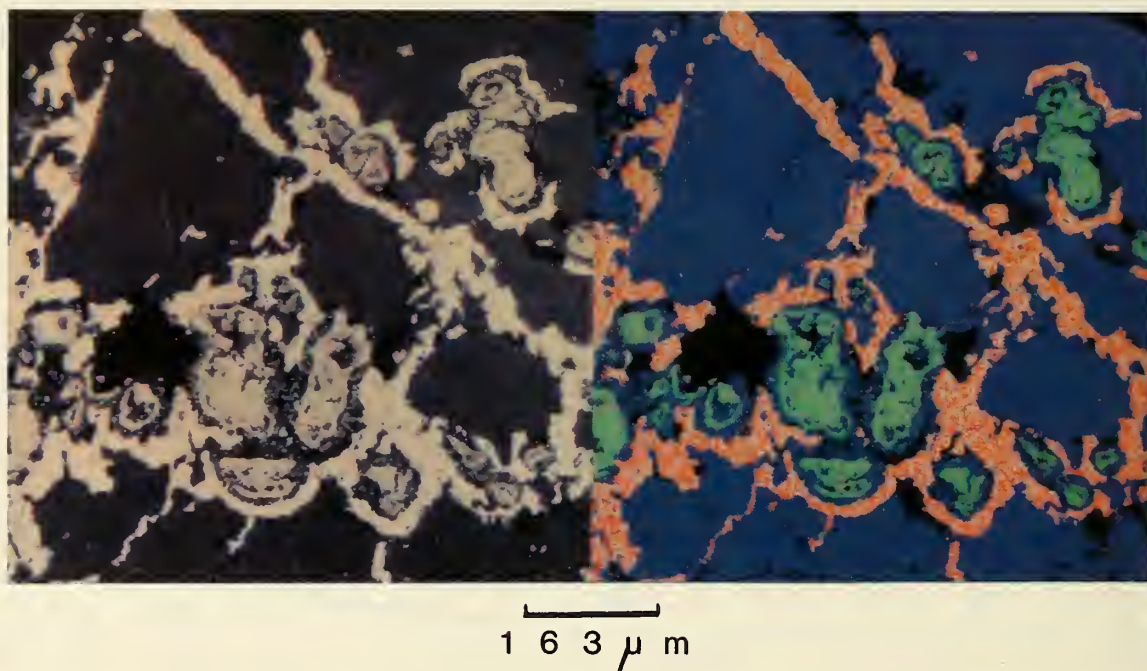


Figure 16.—Left, backscattered electron image (BEI) of fracture-hosted atacamite mineralization (brightest areas) in preleach sample of quartz monzonite ore. Right, BEI image with computer-enhanced X-ray overlay maps that show the distribution of copper (red), iron (green), and silicon (blue). Copper is distributed almost entirely in fractures. The dark-blue areas correspond with quartz, and the green areas correspond with limonite.

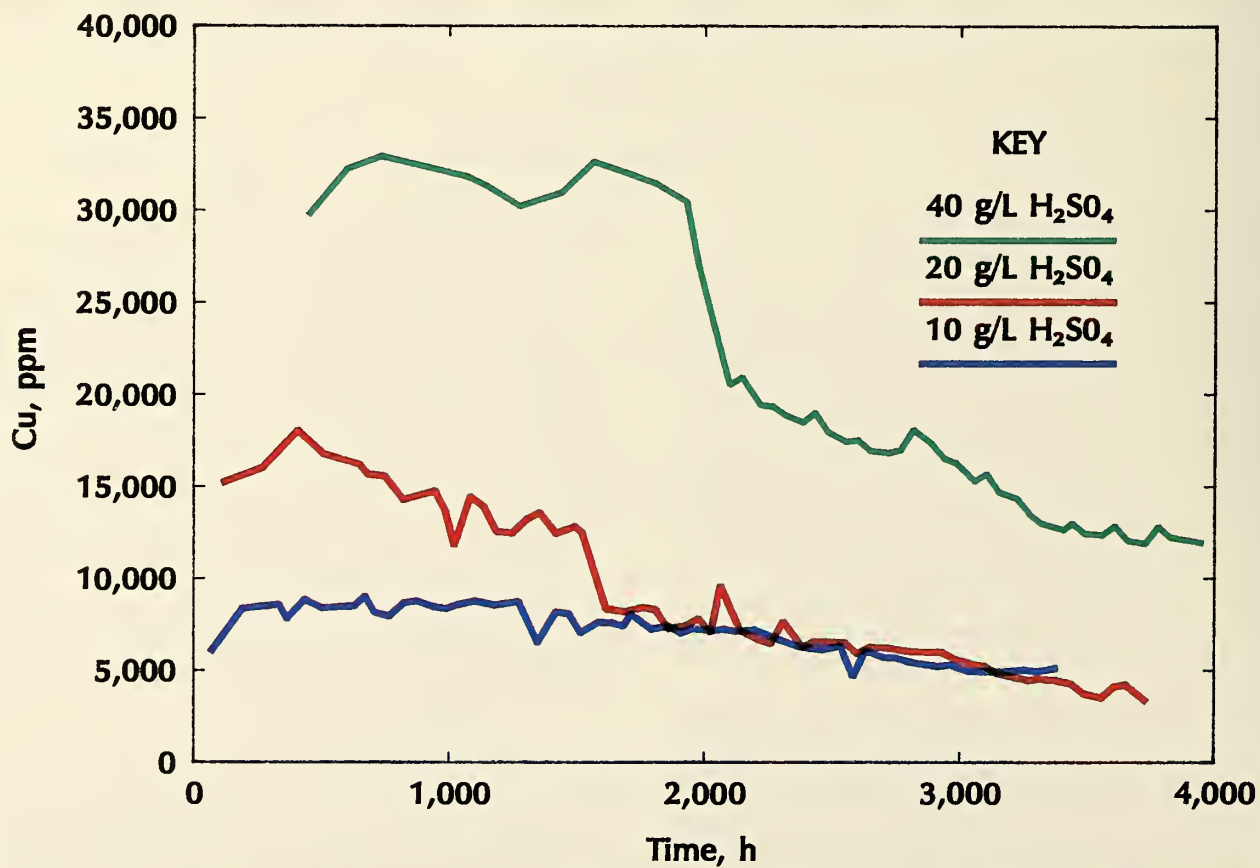


Figure 17.—Copper concentration in solution versus time for three core-leaching experiments using quartz monzonite-hosted atacamite ore.

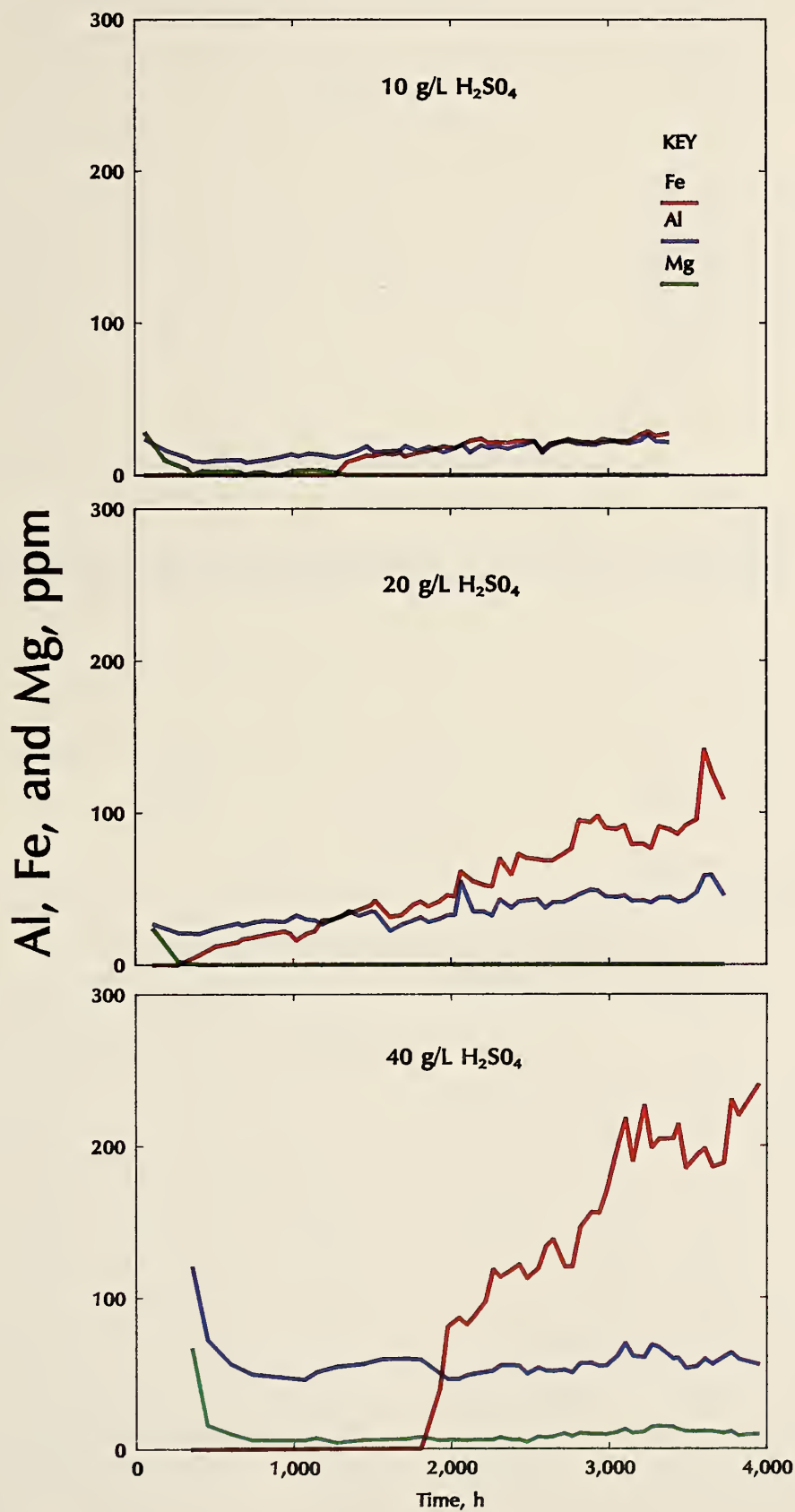


Figure 18.—Aluminum, iron, and magnesium concentration in solution versus time for three core-leaching experiments using quartz monzonite-hosted atacamite ore.

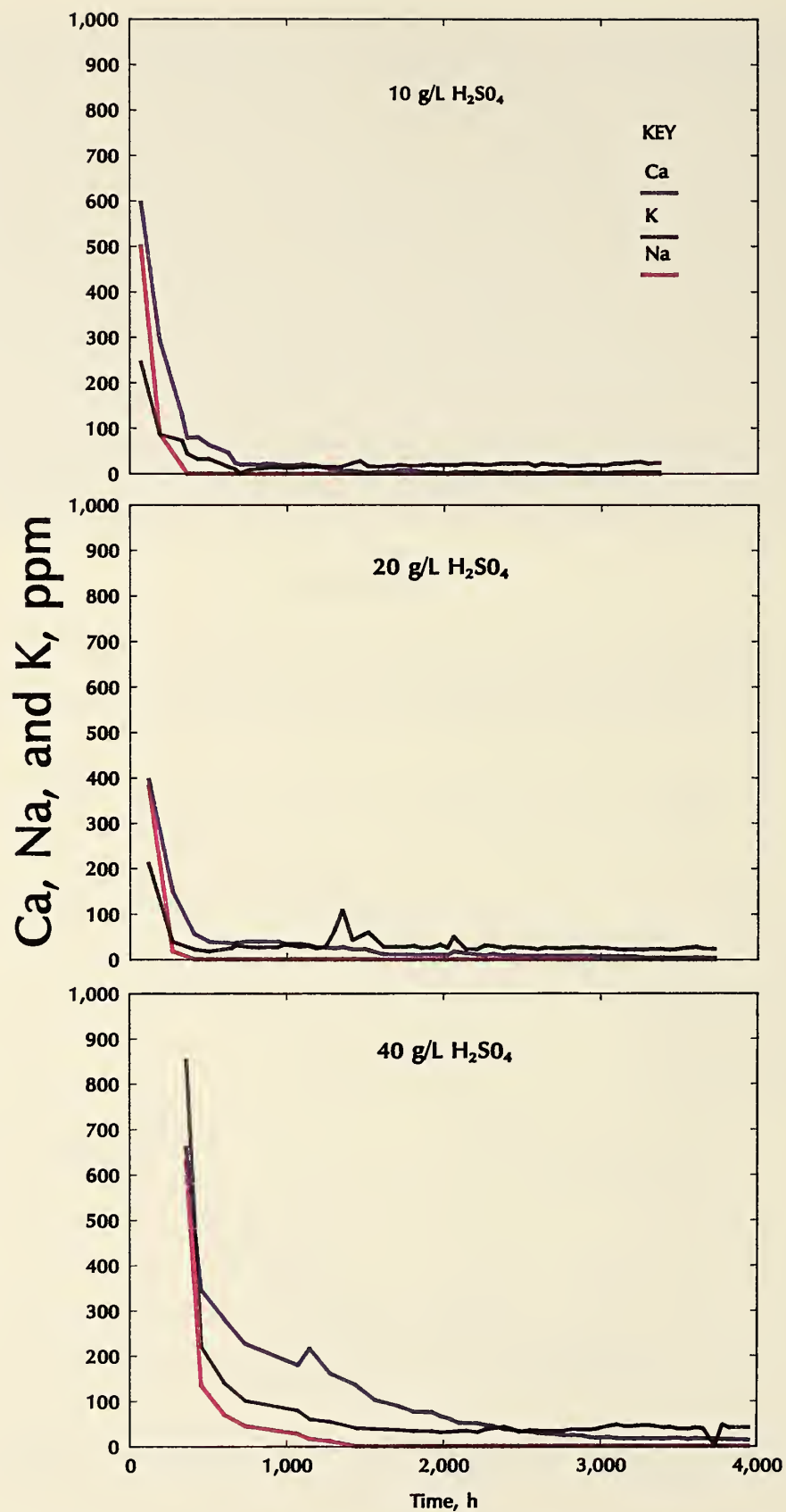


Figure 19.—Calcium, potassium, and sodium concentration in solution versus time for three core-leaching experiments using quartz monzonite-hosted atacamite ore.

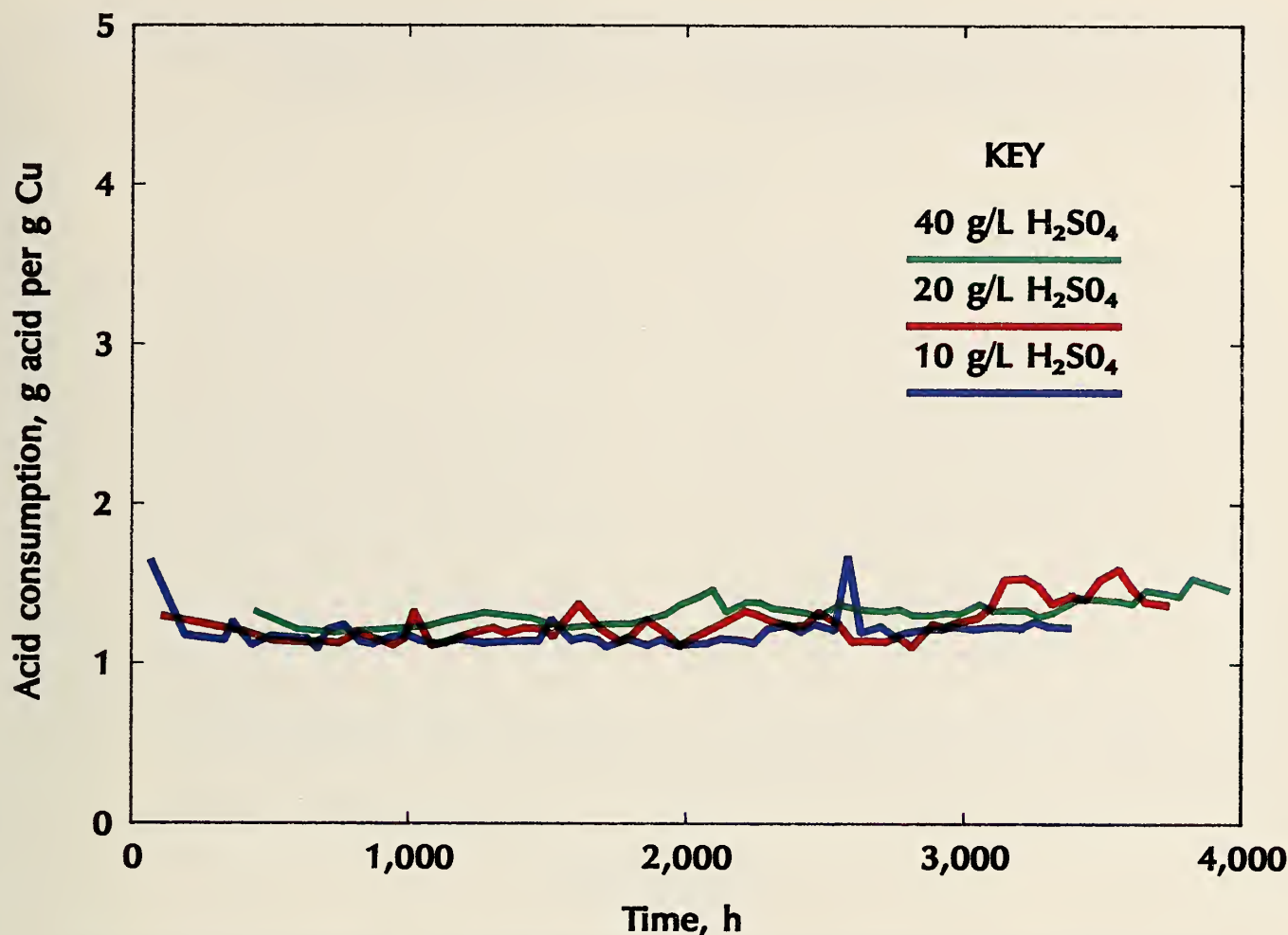


Figure 20.—Acid consumption versus time for three core-leaching experiments using quartz monzonite-hosted atacamite ore.

An important observation is that the acid consumption in these three experiments was relatively independent of acid concentration, unlike acid consumption with the granodiorite porphyry ores. Therefore, a higher acid concentration can be used to achieve greater copper loadings in solution and a greater recovery rate without a prohibitive increase in acid consumption.

As in the experiments conducted on the chrysocolla ore, permeability also increased in all three of the atacamite experiments. The initial permeabilities of these samples were very low, and nearly all copper mineralization was fracture hosted, which led to extremely large increases in permeability relative to the initial values as atacamite was leached from the extensive fracture network. Permeability increased from 100 to 1,300 times in these samples as leaching progressed. It was initially very difficult to inject the leach solution into the samples, but eventually the flow rate increased as atacamite was removed from the fractures. Thus, in situ mining may be applicable to ores that have low initial permeabilities, because mineral dissolution increases the permeability. A small amount of gypsum was observed on the sericite phenocrysts in the postleach samples. However, little SO_4^{2-} depletion was observed in the leachant, indicating that sulfate precipitation was relatively minor in these experiments. The molar ratio of copper to chlorine was near 2 throughout the experiment, indicating that very little, if any, chloride precipitated during leaching.

Applicability of Laboratory Experiments to Deposit Evaluations

Conducting a few core-leaching experiments and combining leaching chemistry data with preleach and postleach petrologic studies of the samples can provide a great deal of information about the applicability of in situ mining for a particular type of ore. Prior to initiating an in situ mining operation, both the types and abundances of gangue minerals present in the various ore types should be identified, as well as their reactivities with the leach solution. This identification will also assist in optimizing the leach solution concentration and will help identify and control potentially detrimental secondary reactions that may occur during leaching. For example, clay minerals can consume large amounts of acid in the exchange of calcium, resulting in gypsum precipitation. Therefore, it may be necessary to treat the ore body with a solution prior to leaching with acid in order to remove the exchangeable clay ions that can have deleterious effects on in situ mining. Alternatively, it may be necessary to avoid leaching certain parts of the ore body in order to eliminate problems that may affect a larger portion of the well field. It is also important to determine the spatial and mineral distribution of copper in the ores. The solubilities of the various copper-bearing phases and the accessibility of leach solution to them will control the total copper recovery and recovery rates. The six

experiments described above resulted in copper recoveries ranging from 57 to 90 pct; leach solution accessibility was the major limiting factor influencing copper recovery. Although copper recovery achieved in the laboratory may be different from that ob-

tained in the field because of scale differences, relatively high recoveries can be attained if an adequate portion of the ore minerals can be contacted by the leach solution.

SUMMARY

The Bureau has developed a comprehensive laboratory research program to study both the chemical and physical influences on in situ mining. The three main areas of this program are core-leaching experiments, monitoring the resulting fluid chemistry from these experiments, and preleach and postleach petrologic studies. Core-leaching experiments more closely simulate the nature of the solution-ore mineral contact in situ than do experiments with crushed rock. Three core-leaching systems are currently being used to conduct experiments on oxide copper ores. The basic design is similar in all three systems; the main differences are in the materials used to construct the reaction cells and the pumps used to deliver the leach solution to the samples. The necessary features of each type of system include flexibility to accommodate a variety of sample sizes and shapes, nonreactivity with the leach solution, relatively low cost, and ability to recover postleach ore samples for petrologic studies. These systems cover the pressure range of 0 to 1,000 psi and flow rates of 2 mL to several liters per day. These systems can also be utilized to conduct leach solution recirculation experiments to study the effects of ion accumulations in the leach solution on the leaching chemistry during in situ mining.

The significance of mineralogy and texture on leaching characteristics was evident when results from experiments using

two distinctly different ore types were compared. A granodiorite porphyry that contained fracture-hosted chrysocolla and disseminated copper in altered plagioclase phenocrysts contained two gangue minerals that contributed significantly to acid consumption. Very high initial acid consumption was due to the exchange of H^+ for Ca^{+2} and Na^+ in the clay minerals. Copper mobilization did not commence until nearly all of the exchangeable cations had been depleted from the clay minerals. The other gangue mineral, biotite, showed appreciable attack by the sulfuric acid leach solutions, resulting in the mobilization of aluminum, iron, magnesium, and potassium. However, postleach petrologic studies determined that little of the copper contained in the biotite was solubilized. Acid consumption was proportional to leach solution concentration. The second type of ore studied was a quartz monzonite porphyry that contained all copper mineralization as fracture-hosted atacamite. This ore was relatively free of acid-consuming gangue minerals such as biotite and clay minerals, resulting in maximum acid consumptions between 1.2 and 1.5 g sulfuric acid per gram of copper. Information such as this is necessary to properly evaluate various ore types for their suitability for in situ mining.

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METHODS FOR DETERMINING THE GEOLOGIC STRUCTURE OF AN ORE BODY AS IT RELATES TO IN SITU MINING

By Linda J. Dahl¹

ABSTRACT

As part of its in situ mining research program, the U.S. Bureau of Mines is studying the geologic structure of the Santa Cruz and Casa Grande West porphyry copper deposits near Casa Grande, AZ. A datum joint set technique was developed to measure joint orientations on unoriented drill core. Based on regional fabric and oriented-core data, a particular joint set is defined as the datum joint set that is oriented in a certain direction. Other fracture orientations are measured relative to this datum joint set. The major joint sets are identified by plotting lower hemisphere Schmidt equal-area projections. This technique allows the structural data base of a deposit to be extended while keeping the high cost of oriented-core drilling to a minimum. Methods of calculating fracture frequency and fracture density are also discussed.

The major joint orientations at the Santa Cruz and Casa Grande West deposits strike northeast and northwest. The most prominent joint set strikes northeast and dips 70° to the northwest with a frequency of 0.76 to 0.98 fracture per foot. The frequency of other identified joint sets ranges from 0.19 to 0.41 fracture per foot.

INTRODUCTION

A thorough study of the geology of an ore body is important in planning and operating underground, open pit, or in situ mining operations. Exploratory drilling provides much of the information necessary to evaluate a deposit for its mining potential. The principal uses of drilling data at conventional mining operations are to calculate ore reserves, predict ground support requirements, minimize the amount of bad ground through which headings must be driven, and minimize water inflow to the mine. Drilling data are also important for planning an in situ mining operation. Since fractures are the primary fluid flow channels during in situ mining, determining the primary joint patterns and preferred orientation of fractures containing ore minerals is important. Previous studies suggest that a preferred orientation of flow is related to primary fracture orientations (1).² This information can be

useful in designing in situ mining well patterns. Modifying well field design (e.g., geometry and spacing) to take full advantage of preferred flow direction and designing injection wells that intersect an optimum number of mineralized fractures may result in increased metal recoveries. Although this has not yet been demonstrated for in situ mining, it will be investigated in future experiments.

The geologic structure of an ore deposit is often determined with the aid of oriented drill core. However, drilling costs are approximately doubled when drilling oriented core. Therefore, the use of oriented core is usually kept to a minimum. Thus, as part of its in situ mining research program, the Bureau of Mines developed a technique to extend the structural data base of an ore deposit at a low cost relative to drilling oriented core, by using datum joint sets.

BACKGROUND

The Bureau is conducting an in situ mining field research project at the Santa Cruz and Casa Grande West porphyry copper deposits near Casa Grande, AZ (figs. 1-2). The Santa

Cruz deposit is owned by ASARCO Santa Cruz Inc. and Freeport Copper Co. The Casa Grande West deposit is owned by Casa Grande Copper Co. The general geology and structure of the Santa Cruz deposit are being studied using exploration drill core. The structural geology of the Casa Grande West deposit has been well documented with oriented drill core (2). To date, there has been no oriented core drilled in the Santa Cruz portion of the deposit.

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²Italic numbers in parentheses refer to items in the list of references at the end of this paper.

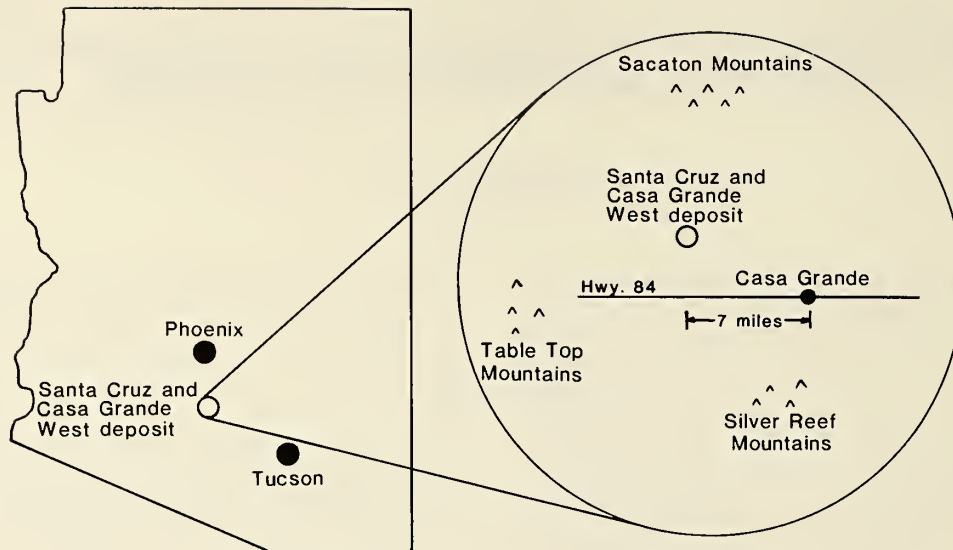


Figure 1.—Map location of Santa Cruz and Casa Grande West ore deposits and surrounding mountains where outcrop data were obtained.

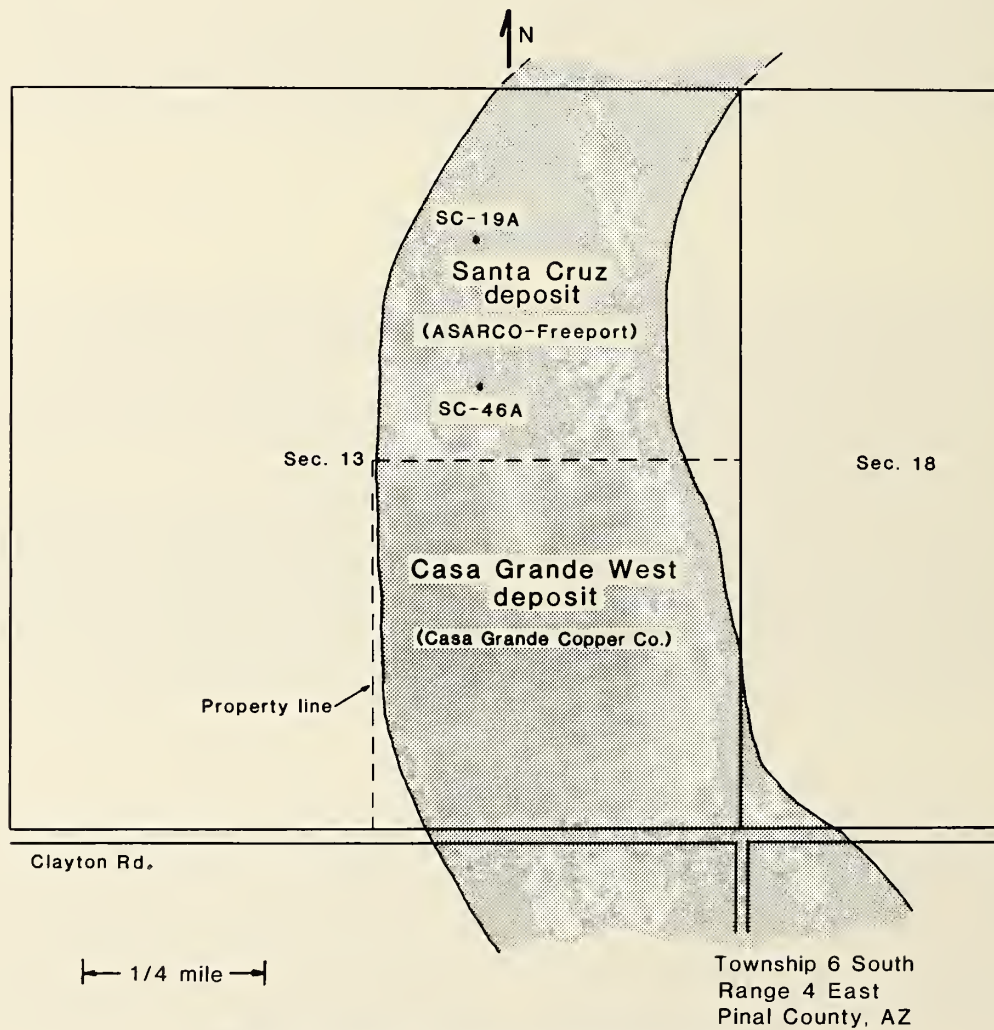


Figure 2.—Santa Cruz and Casa Grande West deposits.

As part of the in situ mining field test program, two new holes were wedged off two previously drilled exploration holes (SC-19 and SC-46) in the Santa Cruz deposit. Unoriented core was obtained near the top of the ore zone in SC-19A (222 ft, from 1,239- to 1,461-ft depth) and SC-46A (275 ft, from 1,410- to 1,685-ft depth).

Holes SC-19A and SC-46A were drilled in a supergene oxide ore deposit. The primary lithologic units are Oracle Granite, quartz monzonite porphyry, and infrequent diabase dikes. Supergene copper mineralization includes chrysocolla, atacamite, and copper clays. Copper mineralization occurs primarily as fracture fillings but also is disseminated.

DATUM JOINT SET TECHNIQUE

HYPOTHESIS

Based on Heidrick's (3) study of porphyry copper deposits in the Southwestern United States and oriented core data from the Casa Grande West deposit (2), areas that have northeast-trending joints dipping to the northwest, it is hypothesized that the steeply dipping joints observed in the Santa Cruz core samples also trend northeast and dip to the northwest. As will be discussed later, this assumption is fully supported by field measurements acquired near the deposit.

PROCEDURE

A joint set dipping 70° to 80° is present in the core. This steeply dipping joint set is hypothesized to strike northeast and dip northwest and is defined as the datum (reference) joint set. At each datum joint, as much surrounding core as possible was reassembled and, where necessary, taped together with masking tape (fig. 3). The drill hole was assumed to be vertical, and dip angles were measured and marked next to each fracture. A mark was made on the datum joint indicating the dip direction. The length of reassembled core was placed in a core tray with the dip direction of the datum joint aligned with the edge of the tray. All the other pieces of core were fit together on either side of the piece containing the datum joint to align them in their original orientation relative to the datum joint.

The core tray edge was used as a guide to mark an "orientation line" along the length of core (fig. 3).

A goniometer is used to measure joint orientations on oriented drill core. Since the core used in this study was unoriented, it was necessary to modify the standard goniometer measuring technique. The goniometer was set at the north 45° east orientation, which is an average orientation of the joints in the area. Core was placed in the goniometer with the orientation line on the core aligned with the goniometer orientation line, ensuring that the primary joint set was dipping northwest (fig. 4). Without moving the core, the goniometer was then rotated to measure the relative strikes and dips of all other fractures (fig. 5).

Many pieces of core were just a few inches in length. To take measurements on these small pieces of core, a 25-cm-long clear plastic extension tube was fabricated and used with wood (25-cm length) and polyvinyl chloride (PVC) spacers (5-, 7-, and 17-cm lengths) to raise and support various sizes of core to the proper height.

ANALYTICAL PRECISION

Dip angles were measured to the nearest 5° . Strikes of 70° - to 80° -dipping joints that were not identified as datum joints had relative measurements, which varied from the datum joint strike by approximately $\pm 15^{\circ}$. Precision in goniometer measurements was approximately $\pm 5^{\circ}$.



Figure 3.—Marking orientation line on reassembled core.



Figure 4.—Aligning datum joint set orientation line on core with goniometer orientation line.



Figure 5.—Measuring relative orientation of fracture.

PRIMARY JOINT ORIENTATIONS

A total of 752 fractures, 383 from hole SC-19A and 369 from hole SC-46A, were measured. The joint orientation data for both holes were plotted on lower hemisphere Schmidt equal-area projections (4). These plots (fig. 6) show orthogonal joint patterns, with the primary joints striking northeast and northwest. Northeast-trending joints dip to the northwest and southeast, and northwest-trending joints dip to the northeast and southwest.

Strike-frequency rosette diagrams were prepared for all fractures (fig. 7), mineralized fractures (fig. 8), and un-

mineralized fractures (fig. 9). There was no significant difference seen between the strikes of mineralized and unmineralized fractures. Approximately 25 pct of all fractures measured contained copper mineralization (atacamite and chrysocolla). The strikes of approximately 60 pct of fractures containing copper mineralization were plotted in the northeast quadrant, and 40 pct of the strikes were plotted in the northwest quadrant. Copper mineralization was observed to be most prevalent in steeply dipping fractures and many near-vertical fractures (fig. 10).

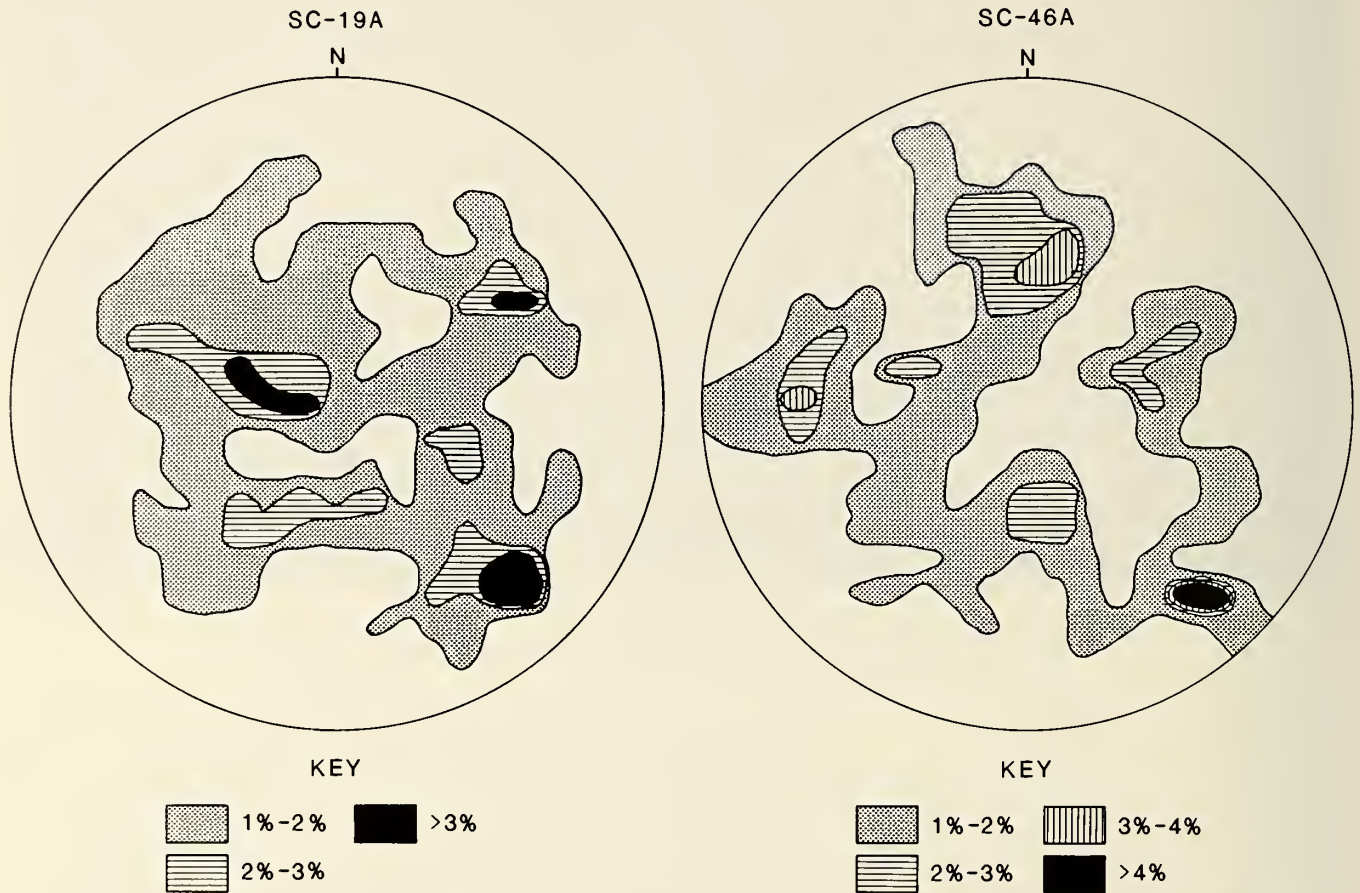


Figure 6.—Lower hemisphere Schmidt equal-area projections for Santa Cruz joints using datum joint set technique. Percentages indicate the number of data points that fall within a 1-pct area of the circle.

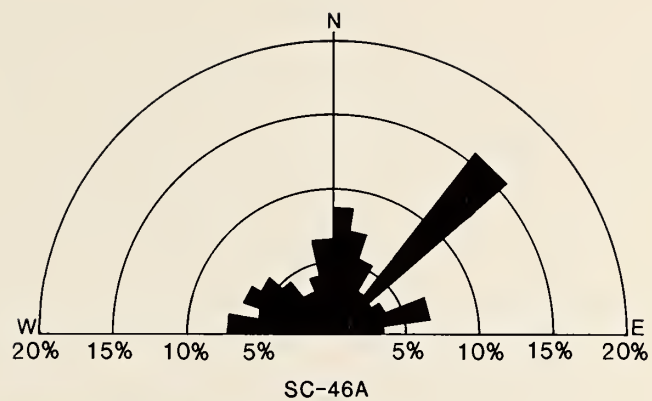
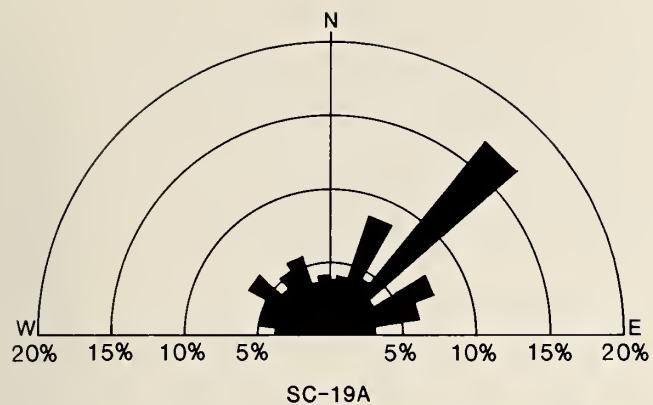


Figure 7.—Strike-frequency rosettes for all fractures, holes SC-19A and SC-46A.

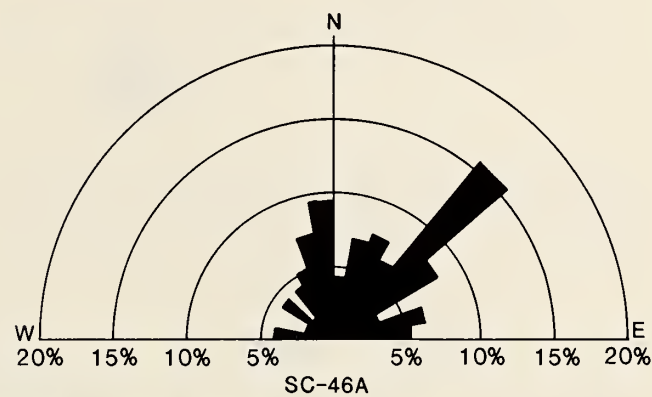
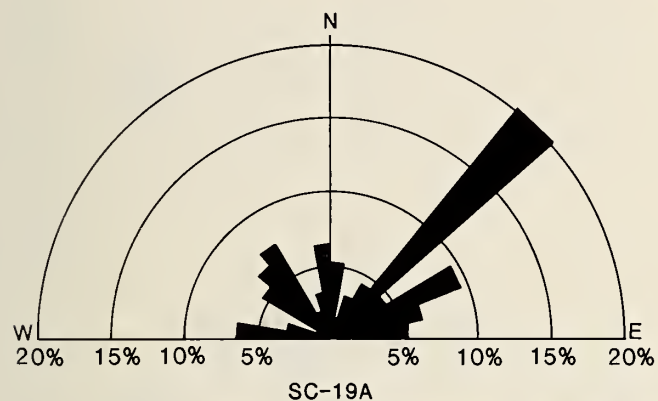


Figure 8.—Strike-frequency rosettes for mineralized fractures, holes SC-19A and SC-46A.

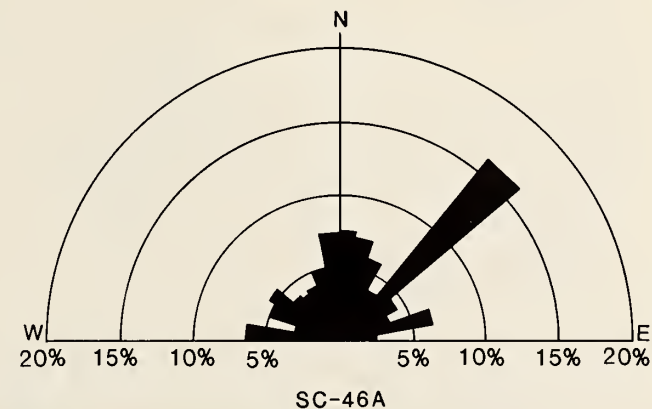
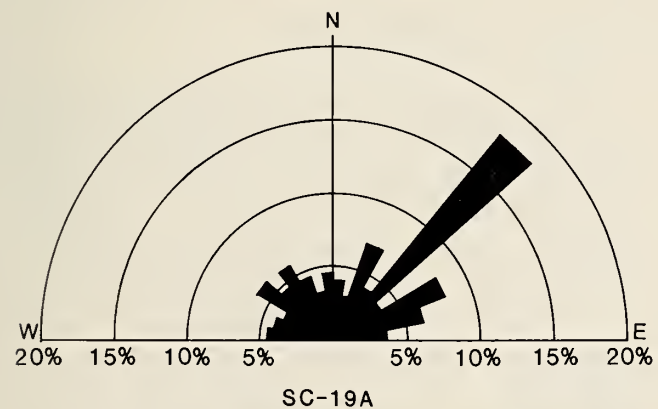


Figure 9.—Strike-frequency rosettes for unmineralized fractures, holes SC-19A and SC-46A.

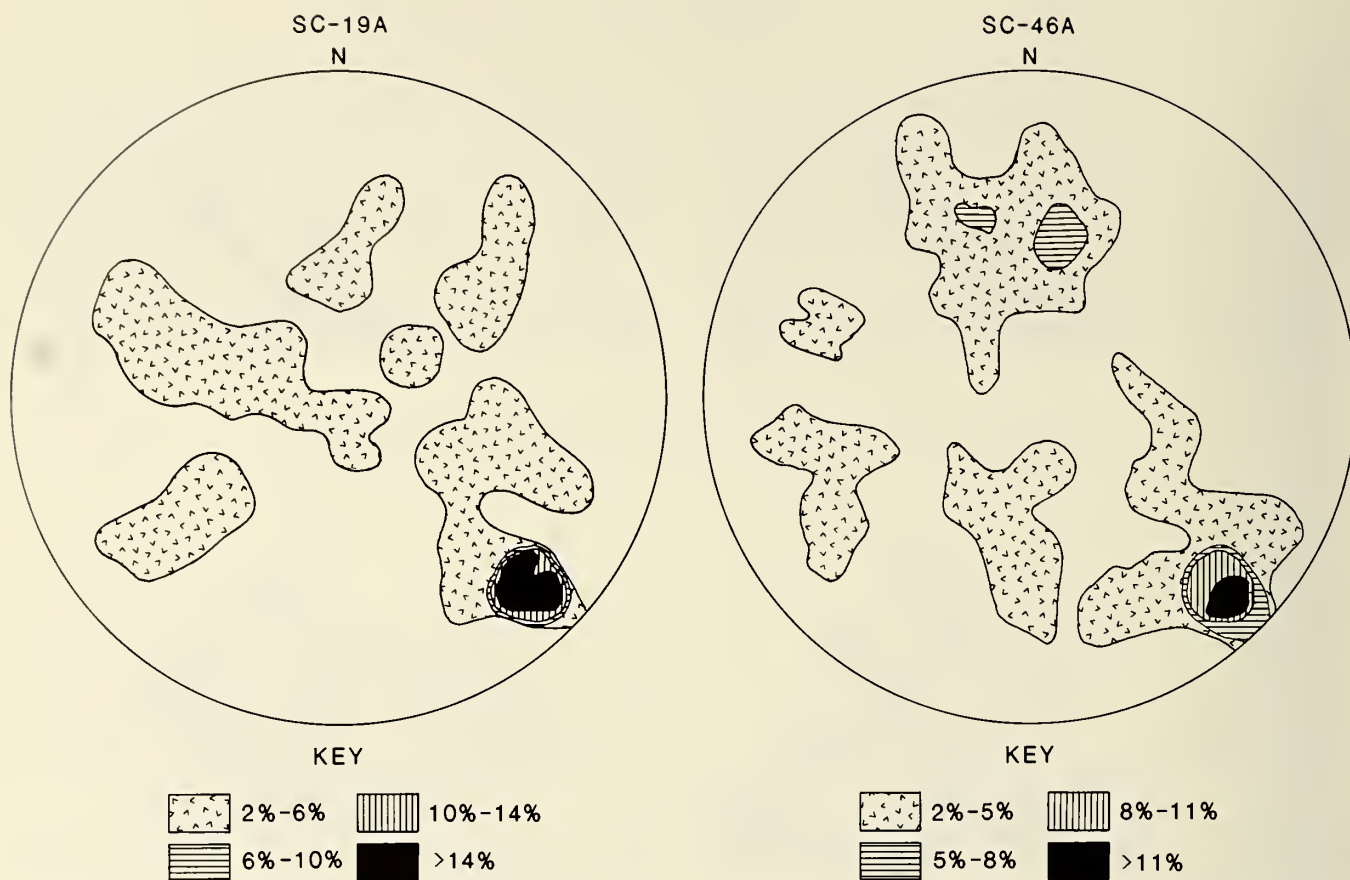


Figure 10.—Lower hemisphere Schmidt equal-area projections for Santa Cruz mineralized fractures. Percentages indicate the number of data points that fall within a 1-pct area of the circle.

VERIFICATION OF DATUM JOINT SET HYPOTHESIS

The joint patterns of the Casa Grande West deposit are well documented from previous oriented-core drilling (2). A composite lower hemisphere Schmidt equal-area projection showing the Casa Grande West joint sets (2) is shown in figure 11. The four predominant joint sets are (1) a northeast-trending set that dips approximately 50° to the southeast, (2) a northeast-trending set that dips approximately 45° to the northwest, (3) a northwest-trending set that dips approximately 45° to the northeast, and (4) a northwest-trending set that dips approximately 40° to the southwest. White (2) did not report many steeply dipping fractures, postulating that this was due to the undersampling of near-vertical joints by a vertical drill hole. This undersampling will be discussed in detail in the next section, "Fracture Frequency."

Table 1 is a comparison of joints identified using oriented core in the Casa Grande West deposit with the joints identified using the datum joint set technique at the adjacent Santa Cruz deposit. The values are averages for all measurements recorded. There is good correlation between the data obtained

Table 1.—Comparison of joint orientation data (average strike-dip) from Santa Cruz and Casa Grande West ore deposits

Deposit	NE-NW	NE-SE	NW-NE	NW-SW
Casa Grande West (2)	45°	50°	45°	40°
Santa Cruz:				
SC-19A	70°	45°	50°	50°
SC-46A	70°	45°	45°	45°

using oriented core and the datum joint set technique. The primary difference observed was that Santa Cruz joints trending northeast and dipping northwest are somewhat steeper than those measured at Casa Grande West. One possible explanation for this difference is that the data collected at the Casa Grande West deposit were from greater depths (approximately 1,800 to 2,950 ft deep) than at the Santa Cruz deposit (approximately 1,240 to 1,685 ft deep). Many of the faults and fractures in this area are observed to be listric in shape, which could account for more shallowly dipping fractures at greater depths.

To further verify the structural fabric of the Santa Cruz deposit, joint orientations were measured at nearby outcrops of the Oracle Granite, which is the same rock unit that hosts the ore deposit. The nearest outcrops of the Oracle Granite are located in the Sacaton Mountains to the north of the deposit, Table Top Mountains to the west, and Silver Reef Mountains to the southeast (fig. 1). Over 700 joints were measured on outcrops surrounding the Santa Cruz Basin, using a Breithaupt Kassel³ geological stratum compass. As seen in the lower hemisphere Schmidt equal-area projections developed from the outcrop data (fig. 12), joint orientations are similar to those observed in the core samples from the Santa Cruz and Casa Grande West deposits. These field measurements verify that the primary joints are steeply dipping and trending northeast and northwest.

³Reference to specific products does not imply endorsement by the Bureau of Mines.

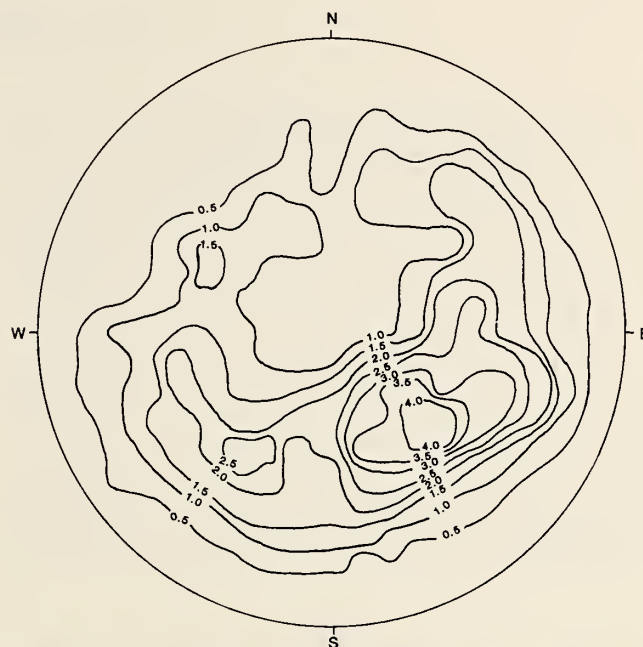


Figure 11.—Lower hemisphere Schmidt equal-area projection for Casa Grande West joints. Contour numbers are percentages that indicate the number of data points that fall within a 1-pct area of the circle. (Based on data from White (2).)

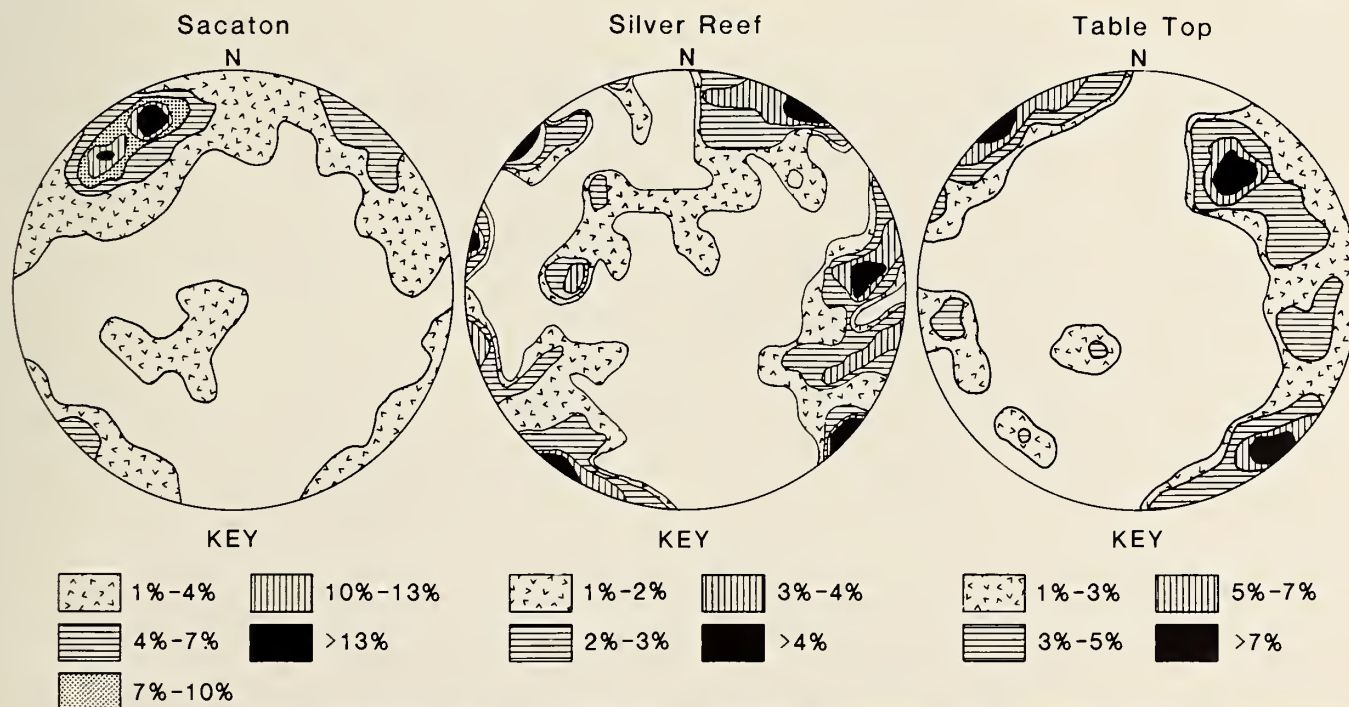


Figure 12.—Lower hemisphere Schmidt equal-area projections for Sacaton, Silver Reef, and Table Top Mountains. Percentages indicate the number of data points that fall within a 1-pct area of the circle.

FRACTURE FREQUENCY

A common technique used to estimate fracture frequency is simply to count the number of fractures intersected in a drill hole and divide it by the drill-hole length. However, these estimates are dependent on the orientation of the observation line relative to the joints. In many porphyry copper systems in the Southwestern United States, the strikes of fractures are not random (3), and a change in observation line can cause large changes in observed fracture frequencies. Thus, there is a natural bias against intersecting joints as the angle between the joints and the observation line decreases. The actual fracture frequency is greatest in a line normal to the joint set (fig. 13). Therefore, fracture frequencies based on the number of fractures intersected in a drill hole are misleading. The following calculations indicate that actual fracture frequencies may be underestimated by as much as two-thirds by the direct observation method. Observed fracture frequency can be converted to actual fracture frequency (number of fractures encountered along a line normal to the dip of the joint set) using the following equation:

$$\text{Actual fracture frequency} = (N_d/L_d)/\cos \theta, \quad (1)$$

where N_d = number of fractures observed in a drill hole,
 L_d = length of drill hole,
 and θ = dip of joint sets.

Actual fracture frequencies were calculated for each joint set, using the data obtained with the datum joint set technique (table 2). The effect of observation line bias is most dramatically seen in the steeply dipping joint set (striking northeast and dipping 70° northwest), where the observed fracture frequency is only a third of the actual fracture frequency.

Table 2.—Fracture frequencies

Hole	Orientation		Fracture per foot	
	Strike	Dip	Observed	Actual
SC-19A	NE	45° SE	0.29	0.41
	NE	70° NW	.35	.98
	NW	50° SW	.22	.34
	NW	50° NE	.20	.30
SC-46A	NE	45° SE	.23	.34
	NE	70° NW	.27	.76
	NW	45° SW	.13	.19
	NW	45° NE	.20	.30

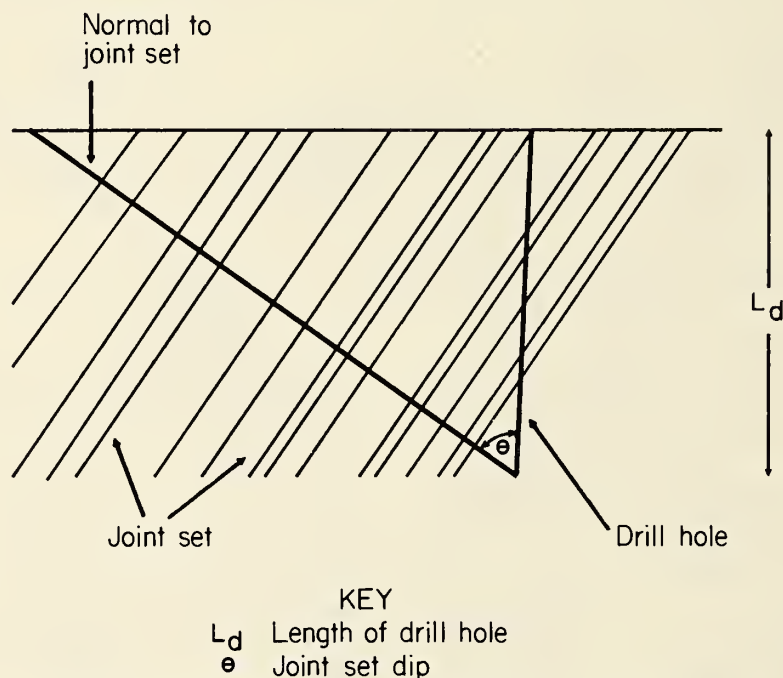


Figure 13.—Undersampling near-vertical joints in a vertical drill hole.

FRACTURE DENSITY

Fracture frequencies calculated with a count of the number of joints in a particular set that intersect a borehole divided by the total borehole length (even those corrected for observation line bias) do not adequately describe possible fluid-carrying surfaces. Fracture frequencies calculated in this way reflect only one set of fractures and not the influence of all the fractures at particular depths. Another way of calculating fracture frequency in drill core is by utilizing the fracture surface area per core volume. Calculations using fracture surface area per core volume will be referred to as fracture densities to differentiate them from fracture frequencies calculated using simple counts of fractures.

Using drill core, fracture densities can be calculated using the quotient of the fracture surface area ($\pi r^2 / \sin \theta_f$) and the volume of length of core ($\pi r^2 L$) (fig. 14) (5). The integrated densities are the sum of these quotients:

$$n = \sum_{f=1}^t \frac{\pi r^2 / \sin \theta_f}{\pi r^2 L} = \sum_{f=1}^t \frac{1}{(\sin \theta_f) L}, \quad (2)$$

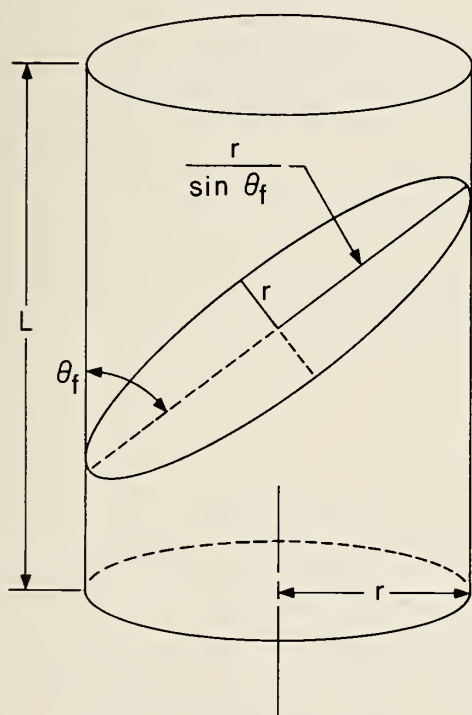


Figure 14.—Fracture density using fracture area per core volume. (Based on data from Haynes (5).)

where n = fracture density,
 f = each individual fracture,
 t = total number of fractures,
 r = radius of drill core,
 θ_f = angle fracture makes with long axis of core,
 and L = length of drill core.

This is a simple calculation using only the fracture angle with reference to the long axis of the drill core and the total length of core. A fracture density plot to overlay geologic cross sections can be generated and contoured to give an indication of highly fractured zones, which may be potential high-flow areas.

Nearly 11,000 fractures were recently measured on drill core to obtain fracture densities for 6 drill holes in the Santa Cruz deposit. Data are currently being compiled, and potential high-flow areas are being contoured. A field test to correlate actual field permeability measurements with those predicted using this method is scheduled for this fiscal year (1989).

$$\begin{aligned} n &= \frac{\text{fracture area}}{\text{core volume}} \\ &= \frac{\pi r^2 / \sin \theta_f}{\pi r^2 L} \\ &= \frac{1}{(\sin \theta_f) L} \end{aligned}$$

where f = each individual fracture,
 n = fracture density,
 r = radius of drill core,
 L = length of drill core,
 and θ_f = fracture angle.

PERMEABILITY DETERMINATIONS USING FRACTURE DENSITIES

One of the variables controlling fluid flow rates is flow porosity or permeability (6). Norton (7) suggests that permeability of a fractured plutonic rock may be approximated by a simple flow porosity model. From Darcy's law for fluid flow through a porous medium, it can be shown that for fractured rocks, permeability is a function of both the abundance of fractures and the cube of the fracture aperture or width (5).

$$k = \frac{nd^3}{12}, \quad (3)$$

where k = permeability, cm^2 ,
 n = fracture density, cm^{-1} ,
 and d = fracture aperture, cm .

Measuring fracture apertures accurately is difficult. Using actual field-measured permeabilities from holes SC-19A and SC-

46A, fracture apertures were calculated using equation 3 to be 7.5 and 4.3 μm , respectively. Since these calculated fracture aperture numbers are very small ($< 10 \mu\text{m}$), they have little effect on equation 3. If the fracture aperture (d) is held constant, permeability calculated using equation 3 is most greatly influenced by fracture density.

Since fractures are the primary fluid flow channels during in situ mining, maximizing the number of mineralized fractures contacted by the leach solutions can result in increased recoveries. Increasing the number of fractures a drill hole intersects increases observed fracture density and therefore permeability. Two of the ways this can be accomplished are by hydrofracturing and by drilling directional holes. Both techniques are costly, but the reward of higher total recovery, greater flow rates, and increased control over solution flow could potentially justify the added expense.

SUMMARY

The Santa Cruz deposit is a highly fractured deposit with primary joints striking northeast and northwest. Orthogonal fracture patterns are observed with northeast-trending fractures dipping northwest and southeast and northwest-trending fractures dipping northeast and southwest. Copper mineralization is most prevalent in steeply dipping, northeast-trending fractures.

The results obtained using the datum joint set hypothesis were consistent with the major joint sets measured using oriented drill core on the Casa Grande West deposit. The results were also consistent with the regional fabric measured on outcrops surrounding the Santa Cruz Basin. The orthogonal joint patterns observed in the Santa Cruz deposit are also typical of porphyry copper deposits in the Southwestern United States as described by Heidrick (3). The technique of

using a datum joint set can be effective in extending the structural data base of a deposit without adding the prohibitive costs involved with extensive oriented-core drilling.

Fracture frequencies based on the number of fractures intersected in a drill hole are misleading. Actual fracture frequencies may be underestimated by as much as two-thirds by the direct observation method. Fracture frequencies are most accurately described along a line normal to the joint set and can be corrected using a simple calculation.

Fracture frequencies do not adequately describe possible fluid-carrying surfaces for qualitatively estimating permeability. Fracture densities calculated using fracture surface area per core volume reflect the effect of all joint sets and allow contouring of potential high-flow areas.

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COMPUTER MODELING APPLICATIONS IN THE CHARACTERIZATION OF IN SITU LEACH GEOCHEMISTRY

By Dianne C. Marozas¹

ABSTRACT

This paper discusses U.S. Bureau of Mines research on using computerized models to characterize in situ mining. Analytical results from Bureau leaching tests were input into the U.S. Geological Survey (USGS) computer program NEWPHRQ, which calculated the component speciation in solution and the stability of mineral phases with respect to the leach solutions. Data from this type of analysis can be used to predict ore solubilities, gangue mineral interferences, and metal mobilities under in situ leach mining conditions. One of the most powerful uses of computer modeling programs, such as NEWPHRQ, is to follow hypothetical reaction paths defined by the user and to calculate the solution compositions that result. Several hypothetical models are presented that demonstrate the potential applications of reaction path modeling to in situ mining. With additional laboratory data on dissolution rates and metastable phase equilibrium, this type of analysis can be used to calculate the metal recoveries expected for a variety of mining situations.

INTRODUCTION

Computer models can be valuable tools for predicting the behavior of complicated in situ leach mining systems, where a large number of variable physical and chemical parameters can affect the efficiency of metal recovery. Bureau of Mines laboratory experiments in leaching whole-core copper oxide ore have provided the preliminary data needed to initiate modeling efforts in the characterization of the chemistry of leaching solutions and in projecting the reaction paths of in situ chemical systems. However, individual laboratory experiments are constrained by the unique variables operating under the conditions of the experiment. Geochemical computer models can extend the predictive usefulness of

laboratory studies by numerically simulating the effects that changing variables will have on leaching results.

The objective of this paper is to demonstrate the range of potential applications of geochemical modeling for practical use in in situ leach mining programs by presenting results from several hypothetical models. The models are designed to test the relative variation in projected metal recovery under a variety of conditions that might be encountered in the field. An accurate computer model for characterizing in situ phenomena is a more efficient way to estimate the effects of a wide number of variables on metal recovery than time-consuming and costly laboratory and field investigations.

BACKGROUND

Computers perform numerical simulations of in situ leaching by solving the set of mathematical equations that describe chemical interactions in a system at equilibrium. Chemical equilibrium models can be used in defining irreversible reaction paths and mass transfer in nonequilibrium chemical systems by assuming that the overall reaction path

can be represented by a series of partial equilibrium states between product minerals and the aqueous phase as the system evolves toward a true equilibrium state. A reaction path model can predict component redistribution and variations in solution composition that result from chemical reactions occurring in rock-water systems. By comparing the model's prediction with analytical data from laboratory experiments, it is possible to evaluate the validity of the reaction model and estimate its accuracy in describing less well-defined systems.

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In recent reviews of the state of the art in geochemical modeling, Nordstrom (1-2)² and Jenne (3) compared the wide variety of computer programs currently available and described the strengths and weaknesses of individual programs in their mathematical construction and in their conceptual approach. Simulation of in situ leaching requires a modeling program that can accept a wide variety of solution components, including leach target metals and related dissolution byproducts that are not usually considered major constituents in natural waters. The program should solve rock-water interactions and simulate irreversible reaction progress in open systems as well as solve for the solution speciation of all components. Kinetic rate constraints derived from laboratory or field data should be easily incorporated into the algorithm for mineral dissolution or precipitation. In situ leach mining systems also require that the selected program be able to handle solutions that range in concentration from initially dilute conditions to highly concentrated conditions. Finally, the computer program should be usable on personal computers in mining offices where mainframe computers may not be available and should allow the user to easily adjust the input so that a number of in situ problems can be modeled.

The computer program selected for initial Bureau efforts in in situ modeling research is the USGS program NEWPHRQ, which is an updated version of computer program PHREEQE (4). NEWPHRQ is a FORTRAN computer program designed to model geochemical reactions based on an ion-pairing model. NEWPHRQ simulates many common hydrogeochemical phenomena such as mixing, rock-water interactions, mineral equilibrium, and ion-exchange reactions. The reaction path, system composition, temperature conditions, equilibrium conditions, aqueous species, and mineral phases are completely user definable, which presents a powerful approach to modeling the unique conditions of in situ mining. NEWPHRQ is currently running on a Bureau personal computer.

POTENTIAL APPLICATIONS OF GEOCHEMICAL MODELING

Geochemical models can characterize the chemical properties of an in situ leach mining system in terms of its dominant variables—major and minor ions, oxidation-reduction potential, acidity, complexing components, mineral stability, and adsorbing surfaces. Computer analyses of these data provide a systematic approach for defining the relative importance of different variables in determining metal recoveries expected during in situ mining.

Computer models can calculate the equilibrium speciation of metals in solution if relevant stability constants are known. This approach presents a distinct advantage over laboratory methods, which are presently unable to determine unequivocal identification of solution species. The term "species" refers to the actual molecular form in which an element is present in solution. For example, copper may exist as one of the free ions Cu^{1+} or Cu^{2+} , or as a complex ion pair, such as $\text{Cu}(\text{OH})^+$, $\text{Cu}(\text{SO}_4)^0$, or CuCl^+ . Speciation can be one of the critical factors in estimating metal recovery because the chemical behavior of any element depends on its chemical form in solution. For example, adsorption of metals onto clay minerals and metal hydroxides can be either reduced or enhanced, depending on what metal complex exists in solution. At high

The PHREEQE family of geochemical models has been successfully applied to a number of hydrogeochemical problems. Plummer (5) used reaction path calculations by PHREEQE in conjunction with mass balance calculations to identify the effects that incongruent dissolution reactions have on ground water compositions within a carbonate aquifer. Marozas (6) used techniques similar to Plummer's that combined PHREEQE's geochemical reaction path modeling with mass balance calculations to study the migration of base metals along ground water flow paths in desert alluvial basins.

In order to extend the application of geochemical modeling to in situ leach mining, several adjustments to the computer program are necessary. The current ion-pairing approach used by PHREEQE and NEWPHRQ is based on the Debye-Hückel mathematical expression for determining activity coefficients of aqueous species appropriate for dilute aqueous solutions. However, Bureau experimental studies show that, when recycling lixiviants, in situ mining will generate highly concentrated solutions, and thus it is important that the program be modified for solutions of high ionic strength. At higher concentrations, the chemical identity of the species begins to play an important role in the thermodynamics of the system. While some of this behavior can be described by empirical modifications to the Debye-Hückel limiting law, such as the Davies equation, which is incorporated in NEWPHRQ, many important contributions are specific to the chemistry of the species. Recently developed phenomenological equations, known as the Pitzer equations, appear to have the required flexibility to describe specific ion interactions and have been successfully applied to a number of geochemical processes (7-8). Pitzer equations can be added to the NEWPHRQ code to accurately model highly concentrated leaching solutions and more closely characterize ore-body interactions with recycled lixiviants.

metal concentrations, the presence of complexes can affect the solubility of solids. Complex formation in solution among constituents derived from dissolving mineral lattices or with constituents of the lixiviant must be taken into account in exploring mechanisms of metal transport by in situ leach mining solutions.

Computer models can test the effects that the variation of system parameters will have on metal recoveries. Acidity effects can be modeled by varying the pH input into the model and charting the consequent changes in metal concentration. Computer models can calculate the variations expected in ore mineral solubilities at different temperatures. Models can simulate temperature variations along flow paths to evaluate potential metal loss that is likely to occur during flow from near the bottom to the surface of recovery wells. Interference from gangue cations on metal recovery can be evaluated by changing input solution composition or increasing the amount of gangue reaction occurring along the reaction path. Kinetic controls on metal recovery can be tested by mathematically varying the dissolution rate of minerals into solution. Initially, the predictive capabilities of computer models can be verified by comparison with laboratory experiments and preliminary field tests; then the model can be extended to numerically simulate recoveries for conditions not duplicated in the laboratory or well controlled in the field.

²Italic numbers in parentheses refer to items in the list of references at the end of this paper.

RESULTS

SPECIATION AND MINERAL STABILITY ANALYSIS

Data from whole-core leaching experiments with sulfuric acid on oxide copper ores from deposits in Pinal County, AZ (9),³ were used to initiate research in modeling in situ leach chemistry. Analytical data from fluids collected at 48- to 72-h intervals were input into NEWPHRQ for each leach experiment. NEWPHRQ was then used to solve for component speciation distribution and for mineral stability during time steps throughout the experiment. The thermodynamic data base used for this analysis is from the compilation of Ball (10).

Typical speciation and mineral stability results will be illustrated by detailed analysis of one of the Bureau's core-leaching experiments in which 1-pct-copper chrysocolla ore from Santa Cruz was leached with 50 g/L sulfuric acid. Figure 1 shows the pH variation versus the amount of solution recovered in the experiment.

Figure 2 shows how the relative concentration of copper species varies as a function of the amount of solution recovered. The amount of solution recovered increases directly with time as sulfuric acid is added to the system. The data show that the speciation and therefore the chemical behavior of copper vary throughout the experiment. Initially, metal sulfate complexes dominate copper speciation in solution; however, as the acidity of the recovered solution increases and about 100 g of solution is recovered, free copper ion becomes the dominant species. This speciation distribution implies that there is less metal complexing and consequently relatively less metal mobility in solution as acidity increases. The causes behind the loss of sulfate complexing can be investigated by analysis of the speciation behavior of sulfate in the system.

Figure 3 is a plot of sulfate speciation in the experiment and shows the relative amount of sulfate that is present as metal sulfate complexes, free sulfate ions, and bisulfate ions. Initially, free sulfate anions and metal sulfate complexes dominate the sulfate system, but with time, acidity increases and sulfate reacts to bisulfate as reaction A is driven to the right.



The fraction of hydrogen sulfate increases at the expense of metal sulfate complexes, and consequently, metal mobility may be reduced. Speciation analyses of particular systems can help determine the lowest acidity required for continued copper oxide dissolution yet maximize metal mobility in solution.

Further information on copper behavior in solution is derived from calculations of the activity or the effective concentration of a species in an electrolyte solution. Evaluation of metal activity in solution is important because it is from this measurement that the solubilities of minerals in the leaching environment can be calculated. The activity of solution components is related to measured molalities by the equation

$$m_S \gamma_S = a_S, \quad (1)$$

where m_S = molality of species S,

γ_S = activity coefficient of species S,

and a_S = activity of species S.

Activity coefficients are determined for each species in NEWPHRQ by the user's choice of either an extended Debye-Hückel formula or the Davies formula. Both of these formulas

calculate the change in activity coefficients as a function of ionic strength.

There is a considerable difference between measured concentrations of total copper and the calculated effective copper

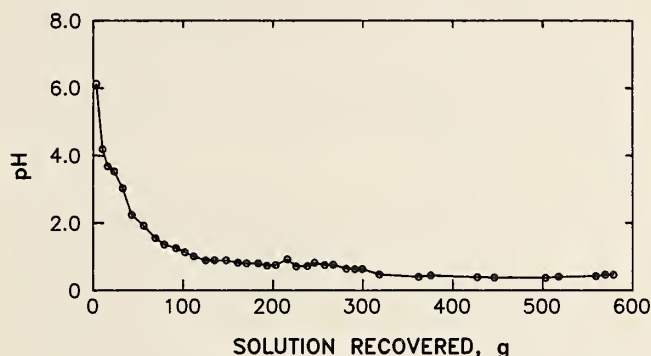


Figure 1.—The pH of leach solutions from 50 g/L whole-core leach experiments on 1.0-pct-copper chrysocolla ore from Santa Cruz.

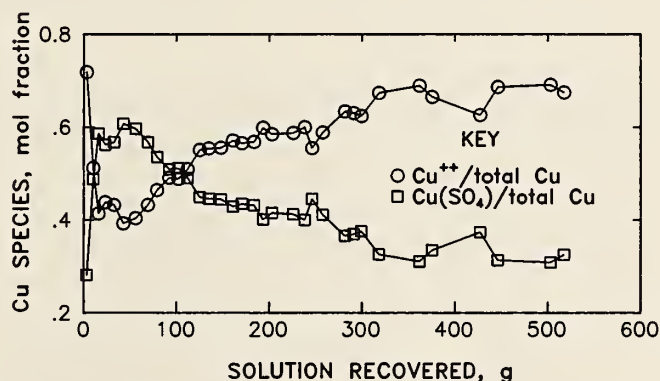


Figure 2.—Mole fraction of copper in experimental leach solutions present as free metal ion and copper sulfate complex.

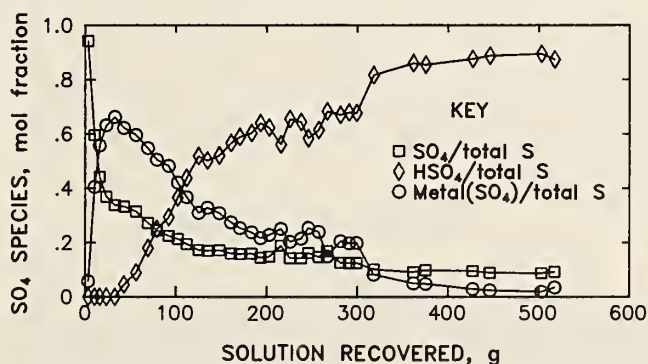


Figure 3.—Mole fraction of sulfate in experimental leach solutions present as sulfate ion, bisulfate, or complexed with cations.

³The deposits are owned by Cyprus Casa Grande Corp. and by ASARCO Santa Cruz Inc. and Freeport Copper Co.

concentration or activity in the leaching solutions from this experiment. Figure 4 shows the total concentration of aqueous copper regardless of its form, which includes free or hydrated copper ion and copper sulfate complexes. The molality and the activity of free copper ion are also shown. The data show that there is a wide variation in total copper concentration, relative to the variation in the activity of free copper. A simple model of copper fluctuation, therefore, may apply to in situ systems that need not account for wide variations in measured copper concentration but rather can focus on explaining the less variable activity of copper. Future research goals of the Bureau are to use computer modeling analyses of the variation in copper activity to provide insights into the kinetic and equilibrium mechanisms that control the in situ leaching of copper.

The activity of copper is emphasized here because it is used for determining the saturation state of mineral phases in the system. The saturation state of an aqueous system, with respect to mineral phases, can be calculated by dividing the ion activity product (IAP) by the solubility product (K_{sp}) constant. The IAP is the product of the activities of the reaction products, each raised to the power indicated by its numerical coefficient, divided by the product of the activities of the reactants raised to the power of their numerical coefficients. The log of the IAP/ K_{sp} ratio is called the saturation index (SI). If a solution is supersaturated with respect to a particular mineral, the SI is positive and the mineral has a tendency to precipitate. If the solution is undersaturated, the SI is negative and the mineral has a tendency to dissolve. If a mineral is at equilibrium, then the SI is zero.

Figure 5 is a plot of SI versus solution recovered for three mineral phases in the Santa Cruz experiment. The SI line for amorphous silica is close to zero or equilibrium throughout the experiment. This implies that the leaching system responds to the addition of silica from the dissolution of chrysocolla or clays by precipitation of amorphous silica. The gypsum SI line is close to equilibrium early in the experiment but falls to undersaturated levels later. This implies that gypsum precipitation would be a problem only in the early stages of in situ leaching. The chrysocolla SI line indicates that it is at first supersaturated but later becomes undersaturated in the experiment. The calculated supersaturation of chrysocolla may have several implications. First, there may be a kinetic inhibition to precipitation during the supersaturation interval, which keeps copper in solution long enough for recovery in the experiment. However, this may not be the case in the field, where time to overcome activation energy barriers to precipitation may be within the residence time of the lixiviants in the host ore. Second, because the activity of copper remains near steady state, this may indicate that there is another, faster control affecting copper concentration in solution other than chrysocolla dissolution, such as the presence of unidentified metastable phases. Finally, the chrysocolla solubility data used in the calculation may need to be revised to account for the solid solution identified by microprobe studies of chrysocolla in the drill core.

APPLICATION MODELS

The most powerful application of computer modeling programs, such as NEWPHRQ, to in situ mining is the program's ability to follow user-defined hypothetical reaction paths and to calculate the resulting solution compositions. Given a set of realistic variables, a reaction path model can predict expected metal recoveries, gangue mineral interferences, and lixiviant losses. Conversely, given unexpected

metal recoveries, reaction path models can test variables that might be causing anomalies and may suggest a means to maximize positive variances and minimize negative variances.

The following sections present several reaction path models that simulate the effect of several variations in system parameters on the leaching of Santa Cruz 1.0-pct-copper chrysocolla ore with 50 g/L sulfuric acid. These models are presented to demonstrate potential applications of computer modeling for in situ mining.

The copper recoveries specified in figures 6 to 13 are calculated for certain theoretical conditions and therefore should not be taken as the copper recovery expected for an actual field situation. Rather, the intent of this study is to show how accurate computer models can be integrated into systematic methods of predicting copper recovery under a given set of conditions.

Effects of Sulfate Complexing on Copper Recovery

Cations in an aqueous medium react with available bases to improve the stability of electrons in their outer shell. Metal ions, such as copper, can react to achieve such stabilization without the formation of precipitates by forming metal coordination complexes with anions or negatively charged molecules called ligands. It is well known that the presence of foreign

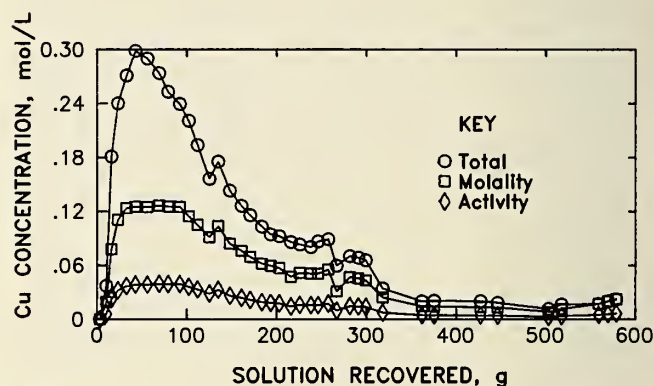


Figure 4.—Concentration of total copper and molality and activity of free copper ion in experimental leach solutions.

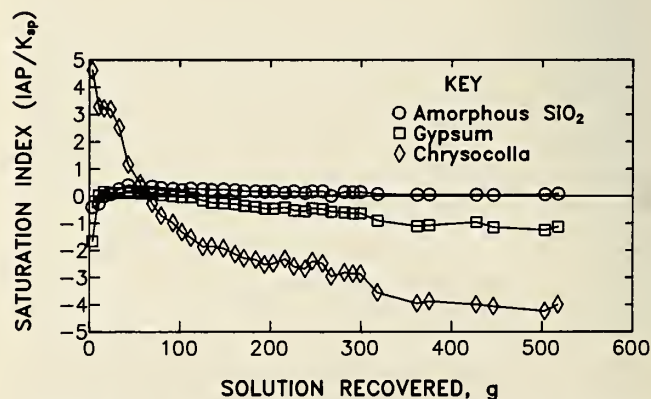


Figure 5.—SI of mineral phases with respect to experimental leach solutions. Zero line represents equilibrium; positive values indicate supersaturation, and negative values, undersaturation.

ligands increases the solubility of slightly soluble metal salts (11). Sulfate in sulfuric acid lixiviants increases the solubility of chrysocolla during in situ mining.

It is possible to illustrate by computer modeling the significance of complex formation in enhancing the solubility of copper solids. In this model, chrysocolla dissolves to equilibrium with solutions of varying acidity, while equilibrium is also maintained with amorphous silica—first, with no sulfate present and, second, with excess sulfate present for complexing. Chrysocolla solubility was estimated from 50-g/L sulfuric acid leach experiments.

Results show (fig. 6) that copper mobility is significantly increased by sulfate complexation in solution. Copper recoveries are approximately $1\frac{1}{2}$ times higher when sulfate is present at a pH of about 0.8.

Effects of Gangue Cations on Copper Recovery

Gangue cations compete with copper for sulfate ligand in solution, and because alkali-earth sulfate aqueous complexes are more stable than copper sulfate complexes, sulfate ligand will preferentially complex with gangue cations, leaving copper in solution as free copper ion. The presence of sulfate-seeking cations from the dissolution of gangue minerals, in addition to copper, has the effect of forcing copper to its free ion state. If free copper is abundant in solution, copper minerals such as cuprite, chalcantite, or native copper may approach saturation levels and precipitate. High concentrations of gangue cations may hinder the transport of copper by forcing precipitation, even at low pH values.

The effects of gangue cations at low pH on copper recoveries were simulated by using NEWPHRQ to calculate the total amount of copper that would enter solution during the equilibrium dissolution of chrysocolla in the presence of gangue cations at various pH levels. Theoretical copper recoveries were calculated for a system with (1) no gangue cations in solution, (2) gangue cations present at concentration levels typical of those found in the late stages of the Santa Cruz experiment (178 ppm calcium, 396 ppm magnesium, 1,090 ppm iron, and 1,390 ppm aluminum) and with 49,000 ppm sulfate, and (3) twice as many gangue cations in solution as those in the Santa Cruz experiments (356 ppm calcium, 792 ppm magnesium, 2,180 ppm iron, and 2,780 ppm aluminum) and 24,500 ppm sulfate. The last model might simulate conditions expected during the recycling of in situ leach solutions.

The results of these simulations (fig. 7) show that gangue cations interfere with copper mobility in solutions during the equilibrium dissolution of chrysocolla in sulfuric acid lixiviant. This model can predict at what level gangue mineral interferences with copper mobility will present an economically unacceptable situation.

Effects of Ionic Strength on Copper Recovery

An efficient way to lower mining costs during in situ mining is to reuse lixiviants that maintain sufficient acidity to dissolve copper oxide minerals. During this recycling, the continuous dissolution of ore and host minerals increases the concentration of solutes in solution and thereby raises the ionic strength of recycled solutions. The effective concentration of metals in electrolyte solutions, and therefore the solubilities of ore minerals, are dependent on ionic strength. As ionic strength increases, solute molecules will, on the average, be closer together and interactions in solution will take place more readily.

Because the activity of copper changes with ionic strength, solubilities will vary with ionic strength. NEWPHRQ was used to investigate the effects of increasing ionic strength on total copper recovery. In order to observe only the effects

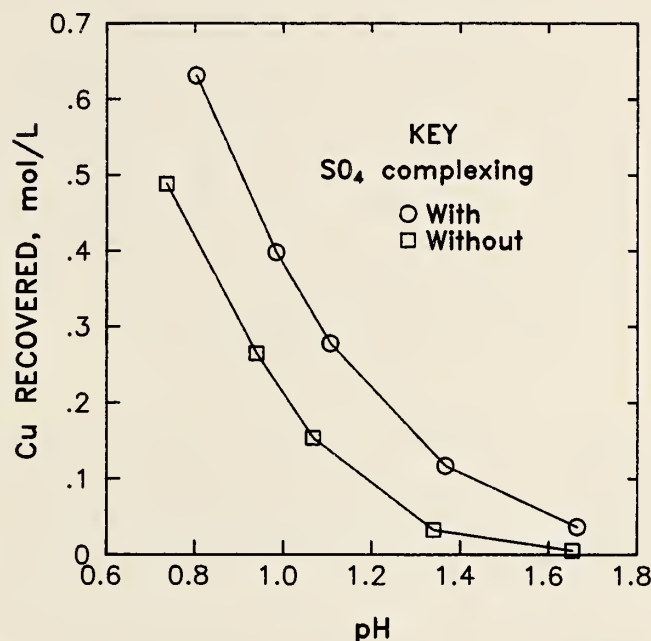


Figure 6.—Equilibrium concentration of copper in presence and absence of sulfate in solutions saturated with amorphous silica.

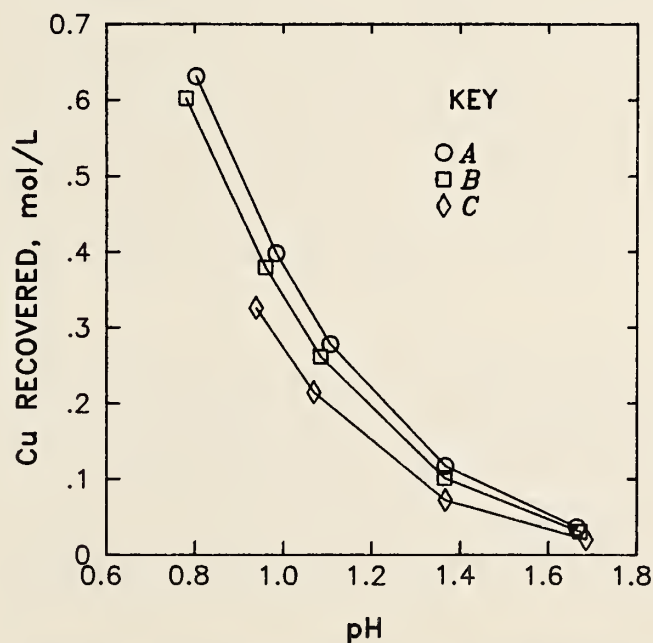


Figure 7.—Equilibrium concentration of copper in solutions saturated with amorphous silica. A, No gangue cations; B, gangue cation concentrations are typical of one-pass leach experiments with 0.54 mol/L sulfate; C, gangue cation concentrations are twice that of one-pass leach experiments with 0.26 mol/L sulfate.

of ionic strength variations without gangue cations or ligand concentration effects as previously considered, ionic strength in the model was changed by the addition of potassium and chlorine, which do not interfere with copper or copper complexes in solution. Potassium and chlorine were added in concentrations sufficient to reach the desired level of ionic strength.

The results (fig. 8) show that, as ionic strength increases at the same pH, copper recovery is decreased. As ionic strength increases, the activity coefficient for copper increases to values greater than one, so that the effective concentration of copper increases. This inhibits the solubility of chrysocolla because saturation is achieved at lower copper concentrations. This model uses the Davies equation for the calculation of activity coefficients, but as was discussed in an earlier section, a specific interaction model would probably describe the solution behavior more accurately at high ionic strengths.

Effects of Increasing Gangue Cations on Clay Mineral Precipitation

Metal cations in solution are known to adsorb onto clay minerals in rock-water systems. Solutions concentrated with gangue cations can approach clay mineral saturation levels even at low pH. When clay minerals precipitate in situ environments, surface adsorption sites are provided for the adsorption and consequent loss of metal from solution.

NEWPHRQ was used to model the saturation state of clay minerals during the concentration of gangue cations. Silica concentrations in solution were held at equilibrium with amorphous silica, and copper concentrations were held constant at 5,000 ppm. Gangue cation concentrations were gradually increased at a system pH of 1.0 and at a system pH of 3.0.

The saturation states of potassium-illite and of magnesium-beidellite minerals in these hypothetical solutions is illustrated by plotting the SI of each mineral (fig. 9). The zero line represents equilibrium; when the SI plots above this line, clay minerals precipitate, while at values below this line, clay minerals dissolve. The results show that potassium clay minerals approach saturation as potassium in solution reaches 0.05 mol/L, regardless of pH. Magnesium clays do not approach equilibrium even at magnesium concentrations of 1 mol/L. Models can predict when gangue cation buildup in recycled solutions might begin precipitating clay minerals.

Effects of Potassium Exchange Reactions on Copper Recovery

Bureau laboratory experiments indicate that a considerable amount of acid is consumed by ion-exchange reactions in the early phases of whole-core leaching. This acid is no longer available to react with copper oxide minerals, which decreases the efficiency of in situ leaching. Reduction of this initial acid consumption may be required to economically leach copper.

NEWPHRQ was used to model the effect of adding potassium to the lixiviant while maintaining a constant number of exchange sites, chrysocolla equilibrium, and amorphous

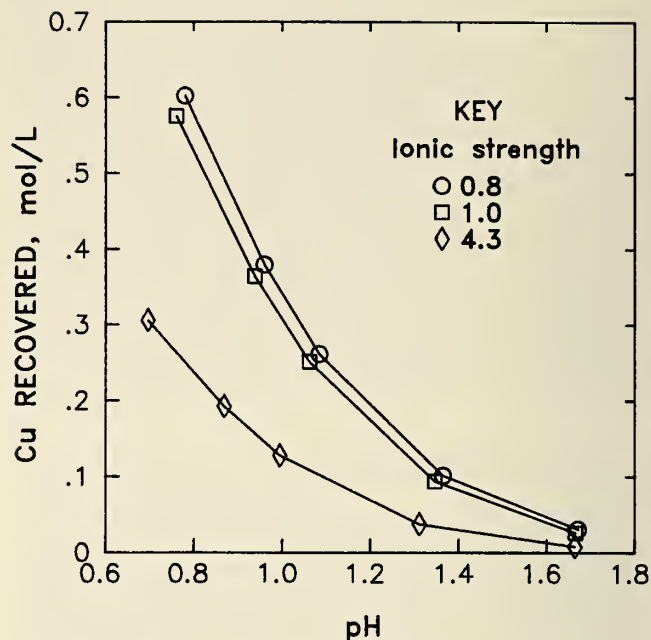


Figure 8.—Equilibrium concentration of copper in solutions of varying ionic strengths saturated with amorphous silica. Zero line represents equilibrium.

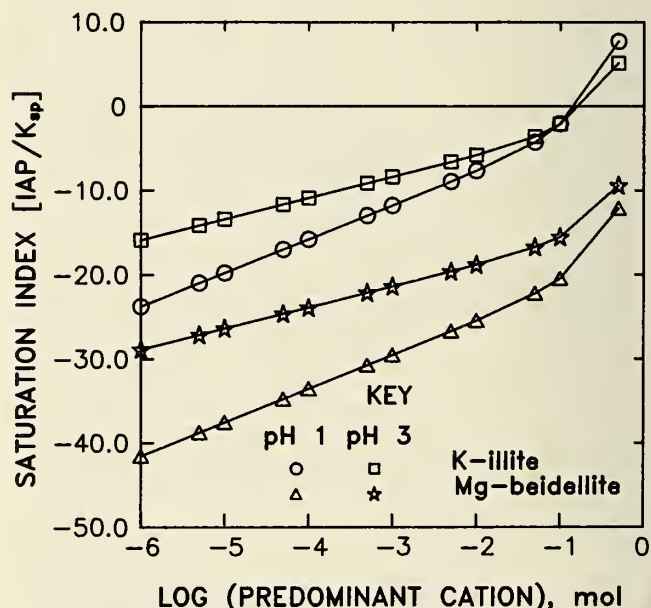


Figure 9.—SI of clay minerals in 50 g/L sulfuric acid, amorphous silica-saturated solution with varying concentrations of gangue cations.

silica equilibrium. The pH of the system is adjusted as exchange and dissolution reactions occur. Potassium affects the reaction by limiting the amount of cations exchanging for hydrogen ion at clay exchange sites. Consequently, acid loss from exchange reactions is diminished and chrysocolla dissolution is enhanced.

Figure 10 is a plot of copper recovered from the leaching of chrysocolla ore as a function of potassium concentration in the lixiviant. Copper recovered without added potassium is shown by the "potassium-free" line. This figure shows that significant differences in copper recoveries are expected only when potassium exceeds 0.01 mol/L in the solution. The economic feasibility of adding potassium or other outside reagents to lixiviants for copper recovery enhancement can be determined by computer estimations of how much of the reagent will be required to significantly increase copper recoveries.

Effects of Biotite Dissolution on Copper Recovery

Characterization and leaching studies on Santa Cruz ore (9) have shown that the presence of biotite in the rock matrix inhibits the leaching of copper. Biotite dissolves in acid, which removes acid from the lixiviant, increases the pH, and promotes copper mineral precipitation. Gangue cations released by biotite dissolution compete with copper for sulfate ligand and consequently can inhibit copper mobility in solution.

Biotite interference in copper leaching can be simulated with computer models. The model simulates the dissolution of biotite into a system in which equilibrium is maintained with chrysocolla and amorphous silica at low pH. The amount of biotite that dissolves is increased throughout the model to gauge the effects of increased biotite reactivity in the system.

Figure 11 shows that copper recovery decreases by an order of magnitude as 0.15 mol of biotite dissolve. The extent of biotite mineralization in potential ore bodies can be estimated by petrologic examination and the consequent effects of biotite reactivity on copper mobility projected by computer modeling.

Effects of Copper Dissolution Rates on Copper Recovery

Characterization work on thin sections of oxide ore deposits (9) has revealed that copper is present in two different mineral phases—chrysocolla and copper-bearing clays. Chrysocolla is principally fracture hosted, while copper-bearing clays are principally disseminated in the rock matrix. The amount of copper released by solubilization of copper ore depends on the dissolution kinetics of the two different phases. Laboratory leaching studies on this type of ore show that copper recovery reaches a maximum in the early stages of leaching (fig. 12) and then decreases to a near-constant level for the remainder of the experiment.

Path-dependent reaction progress models are simulated by NEWPHRQ as a series of steps where the resulting solution from one step becomes the starting solution for the next step of the model. NEWPHRQ can simulate expected copper recovery from two different mineral phases by varying the rate at

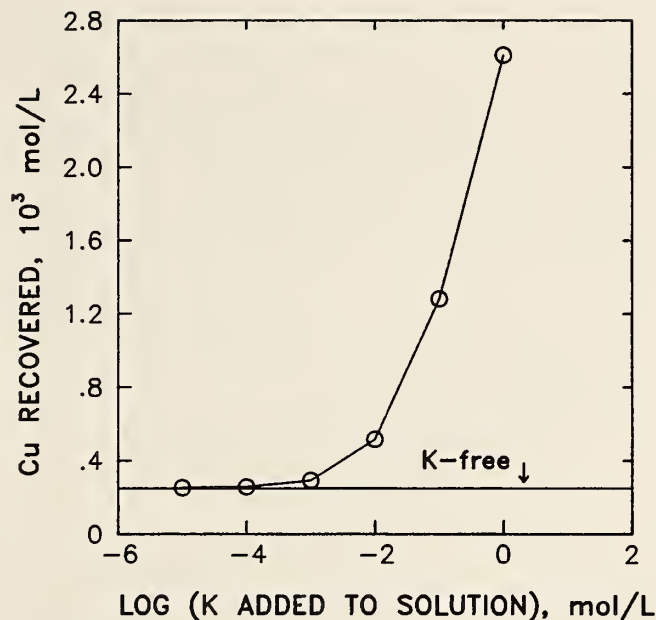


Figure 10.—Equilibrium concentration of copper in 50 g/L sulfuric acid, amorphous silica-saturated solution as function of potassium concentration in lixiviant. "Potassium-free" line marks copper concentration level when no potassium is present.

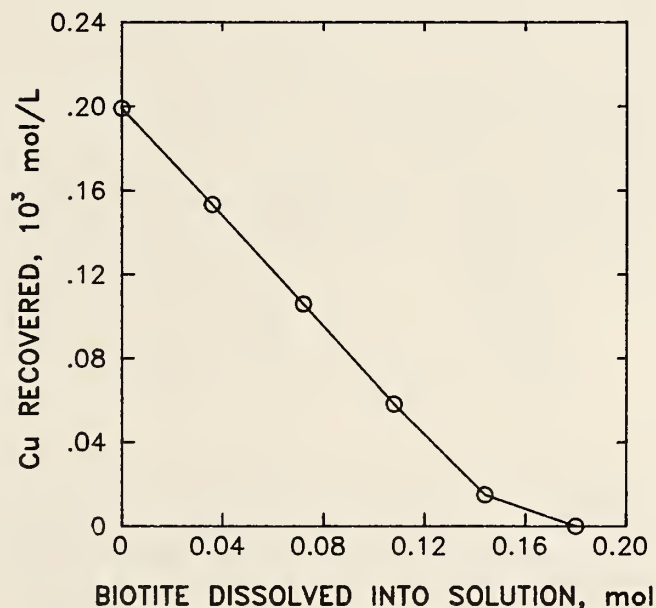


Figure 11.—Equilibrium concentration of copper in 50 g/L sulfuric acid, amorphous silica-saturated solution as function of increasing biotite reactivity.

which each phase is added to the system, allowing the solution to come to equilibrium with amorphous silica and passing the solution on to the next step. Depletion of fracture-hosted chrysocolla is simulated by stopping the chrysocolla dissolution reaction while maintaining the matrix clay dissolution at the same rate as before. Acid is added at each step to simulate the continual influx of acid to the system.

The results of the model are shown in figure 12, where a peak similar to that observed in the laboratory experiment is generated. The results of this model suggest that differing dissolution kinetics of copper-bearing phases are potential controls on copper recoveries in the experiment. This example illustrates one way that computer simulations may explain the reactions responsible for copper release and subsequently can predict expected copper recoveries.

Effects of Temperature Variations on Copper Recovery

Temperature variation during in situ mining may affect copper recoveries. Temperatures may vary considerably when ore deposits are leached at depth in areas with high thermal gradients. The basic relationships that describe the influence

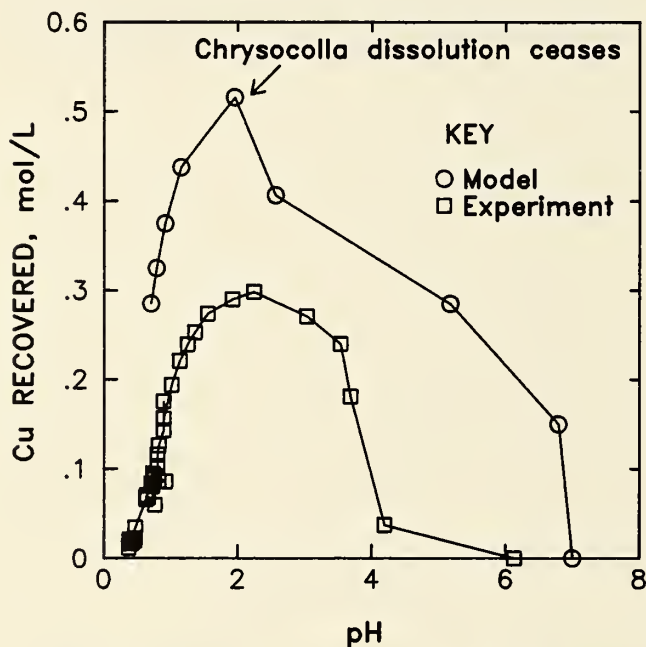


Figure 12.—Effects of two different rates of ore mineral dissolution on copper concentration. Experimental data are from 50 g/L sulfuric acid leach experiments on 1-pct-copper chrysocolla ore. At the point indicated on the diagram, chrysocolla dissolution ceases while copper-clay minerals continue to dissolve.

of temperature on the equilibrium constant of a chemical reaction or phase equilibrium require knowledge of the enthalpy of reaction, the heat capacity of the reaction, and the variation of the heat capacity with temperature. Heat capacity data on chrysocolla solid solutions are limited or unavailable, and enthalpy data must be estimated. Until more accurate data are obtained from direct calorimetry and by measurement of equilibrium properties of chemical reactions, predictions of copper recovery by chemical models are necessarily hypothetical. The following is presented only as an example of how models can calculate copper recoveries when thermodynamic values for the mineral phases of interest in the system are assessed.

Computer simulations are especially useful for modeling changing temperature conditions because of the complexity of the effects of temperature on the entire system. In general, increasing temperature favors dehydration reactions, solubilization of chrysocolla, and ion association reactions, while acid dissociation diminishes.

The model used to demonstrate temperature variation effects on in situ mining was designed so that chrysocolla dissolves to equilibrium as the pH varies and the system is maintained in equilibrium with amorphous silica. Figure 13 illustrates how temperature can affect copper recovery when calculated with an estimated chrysocolla enthalpy. Results show that at similar acidity levels more copper is recovered at 100° C than at 25° C.

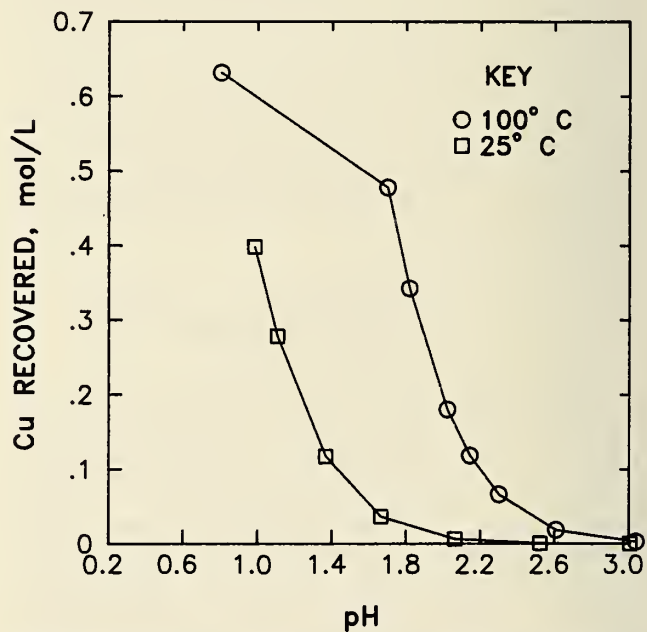


Figure 13.—Equilibrium concentration of copper in 50 g/L sulfuric acid, amorphous silica-saturated lixiviant at 25° C and 100° C.

SUMMARY

It has been demonstrated that computer modeling programs have applications for in situ mining through the quantification of factors that affect the leaching chemistry. To estimate ore recovery and lixiviant losses, knowledge of ore mineral solubility and in situ fluid-rock interactions is essential. Geochemical models can characterize the chemistry of in situ systems in terms of the speciation of solution components and can identify those mineral phases that should dissolve and those that should precipitate under leach mining conditions.

Several models have been developed to demonstrate how computer simulations can be used to predict ore recoveries for a variety of conditions not evaluated by laboratory testing. These hypothetical models calculate the effects of different copper dissolution rates on the behavior of copper recovery curves, the effects of preflushing with potassium to diminish hydrogen exchange, and the effects of temperature on chrysocolla solubility.

Although it is generally known that highly reactive gangue constituents in the host matrix can consume acid and raise mining costs, mathematical treatment of gangue reactivity can project acid consumption and establish at what gangue concentrations it would no longer be economically feasible to leach-mine. Several different models that address the impact of secondary reactions on copper recovery show the wide variety of approaches to this problem that computer simulations make possible.

The models presented in this paper are initial attempts to use computer simulations for predicting copper behavior during in situ leaching. Future work by the Bureau will focus on continued refinement of the geochemical models by calibration of the data base with experimental results, by the incorporation of known kinetic constraints into the program, and by testing of the program under actual field conditions.

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MODELING INFILTRATION TO UNDERGROUND MINE WORKINGS DURING BLOCK-CAVE LEACHING

By Robert D. Schmidt¹

ABSTRACT

The U.S. Bureau of Mines has modeled the infiltration of leach solution to an underground mine void during block-cave copper leaching. The underground leachant collection area is represented as a distributed hydrologic sink, referred to as an "area sink." The area sink solution, which was developed previously using the analytic element method, is described briefly. The area sink model is calibrated against field data from well "drop" tests and an in-mine flow survey; it is then used to estimate the size and shape of the free saturated plume of leachant that exists above the mine void during leaching. An inverse problem is posed, using area sink and well solutions to estimate the leakage coefficient and permeability of ore material above the mine during ongoing leaching operations.

INTRODUCTION

Understanding and controlling the distribution of leach solutions in unsaturated ore material is essential for conducting safe and efficient leaching operations above the ambient water table. The development of a distributed area sink solution for modeling infiltration to underground mine workings during block-cave leaching is part of a Bureau of Mines effort to provide the industry with modeling tools that facilitate understanding of in situ leaching hydrology in both saturated and unsaturated flow settings.

A distributed area sink solution, developed using the method of analytic elements and previously used to model localized aquifer infiltration from surface sources such as ponds, rivers, and lakes, has been adapted for modeling infiltration to irregularly shaped underground mine workings.² The solution, which is highly efficient, has been implemented in a Bureau hydrology computer program, MINEFLO. This paper describes the application of the area sink solution to the hydrologic setting of a block-cave leaching operation near San Manuel, AZ.

METHOD OF BLOCK-CAVE COPPER LEACHING

Block-cave copper leaching is a method by which copper is leached in situ from low-grade ore that has been at least partially rubbled by prior conventional block-cave mining activities. In one variation, shallow wells are constructed on the surface of a rubbled oxide ore mass and an acidic leach solu-

tion is injected into the wells. The leach solution flows downward through the rubbled mass until it reaches an existing underground mine drift. The mineral-bearing solution is then collected in the mine workings, pumped to the surface, and processed to extract metal values.

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²Strack, O. D. L. *Groundwater Mechanics*. Prentice-Hall, 1987, 643 pp.

Figure 1 is a schematic cross section showing the setting of a block-cave copper-leaching operation near San Manuel, AZ. The entire setting is situated above the local ambient water table. The rubbled zone in figure 1 is low-grade oxide ore that was left behind following conventional block-cave mining of an underlying sulfide ore. The thickness of the rubbled ore mass in figure 1 averages 1,626 ft. Injection wells are drilled from the surface to an average depth of 910 ft. The bottom 200 ft of each well is slotted pipe. The underground mine workings in figure 1 consist of an undercut drift at elevation 1,125 ft mean sea level (msl) and a haulage drift 65 ft below it, at elevation 1,060 ft msl. The haulage level and the undercut level are

connected by vertical transfer raises. The underground workings that serve as a collection drift for injected leach solution are on the haulage level.

The cascade area in figure 1 is a zone of partially rubbled ore, which does not directly overlie the undercut workings. Ore material in the cascade area is characterized by mining-induced fractures or crack lines, which run parallel to the boundary of the undercut workings and dip at an angle of about 50°. There is no evidence that these fractures intercept the haulage workings. Thus, there are almost no permeable connections between the cascade area and the underlying mine.

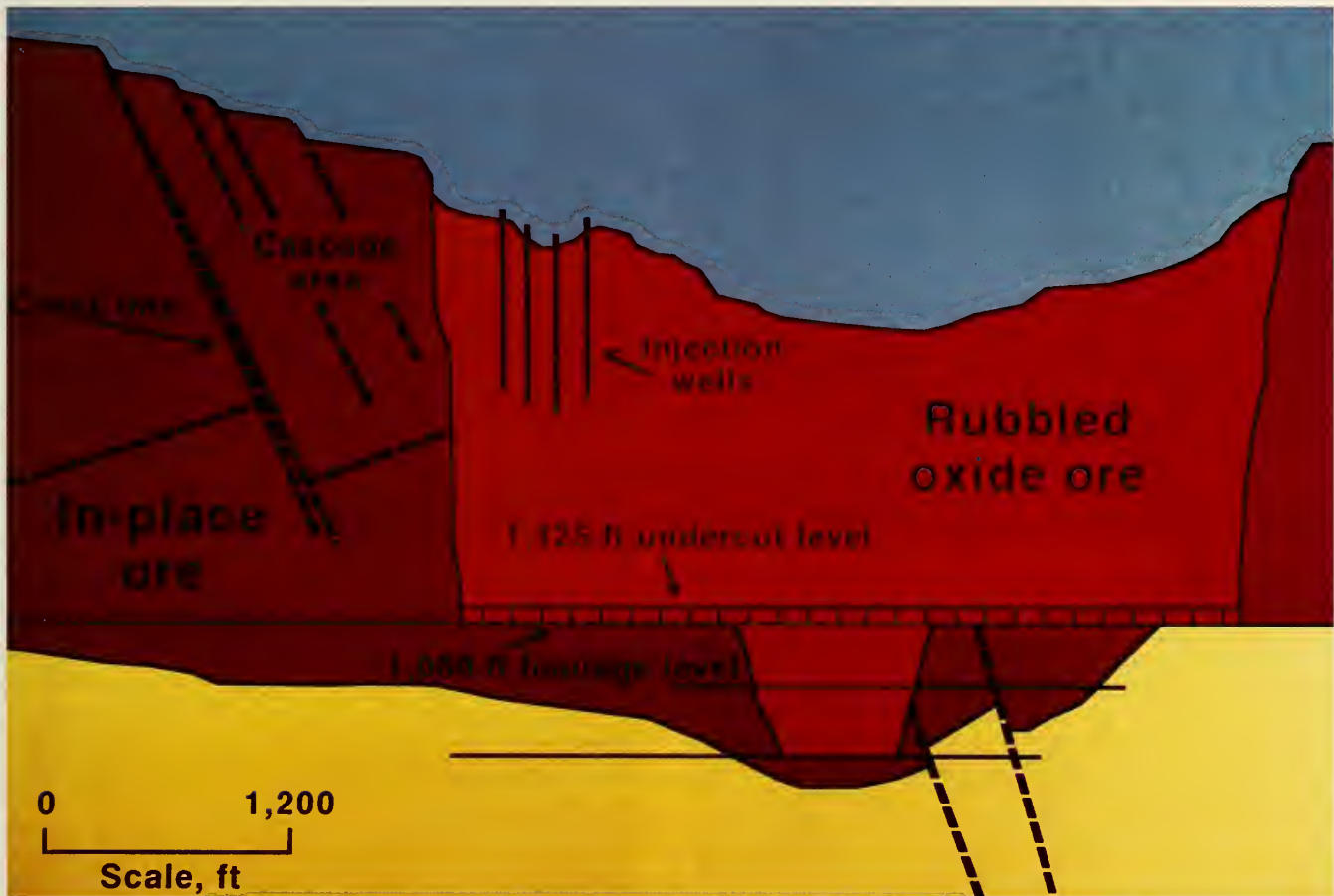


Figure 1.—Schematic cross section of block-cave leaching.

AVAILABLE HYDROLOGIC FIELD DATA

At the time of this study, block-cave leaching operations had been ongoing at this site for about 72 weeks. Field data that describe the hydrologic setting during a 16-week interval of the mining operation, from weeks 26 to 41, consist of the following: (1) a daily history of the total solution injection rate in wells and the total solution recovery rate inside the mine, (2) an underground survey of mine inflow conducted during week 41, and (3) the results of steady-state "drop" tests conducted in three wells during week 26. These data are used to calibrate a model of saturated conditions at the site.

INJECTION AND RECOVERY HISTORY

The total daily rates of solution injection and recovery during the first 72 weeks of leaching operations are plotted in figure 2. Variation in total injection rate during this period was due mainly to variation in the number of operating wells. The operational life span of injection wells averaged about 24 weeks. Well losses occur as a result of subsidence and plugging.

Fourteen wells were in operation in the interval from weeks 26 to 41. As figure 2 indicates, the injection and recovery rates during this period were relatively stable; injection averaged 0.85 million gal/d, and recovery averaged 0.4 million gal/d. During this period, about 0.45 million gal/d of injected solution was going into phreatic (free-surface) storage; thus, the saturated plume was expanding.

MINE INFILTRATION SURVEY

An in-mine survey to locate the transfer raises where leach solution was either flowing or dripping into the mine was conducted on the haulage level during week 41. Figure 3 is a plan view of the mine workings, showing the results of this survey. The outlines of the undercut workings and the haulage workings are shown in this figure in relation to the 14 overlying injection wells. The locations of 94 flowing and dripping transfer raises are represented on this figure as black rectangles. Dry transfer raises are not marked. The surveyed area of inflow in figure 3 is called the in-mine collection area. Because of the low permeability of undisturbed ore, there was no measurable infiltration to haulage workings located beneath the cascade area, northwest of the undercut drift in figure 3.

The steady infiltration of leach solution observed in the 94 transfer raises indicates that saturated conditions had developed in a substantial portion of the overlying rubble material owing to 41 weeks of continuous solution injection. Further, the locations of flowing and dripping transfer raises south and east of the injection wells in figure 3 suggest the approximate location of the saturated boundary as it exists above the undercut workings, during week 41.

WELL DROP TESTS

Also shown in figure 3 are the locations of three wells in which drop tests and permeability slug tests were conducted. During these tests, wells 38, 42, and 45 were shut down one at a time, so that 13 injection wells remained in operation.³ The decline in solution level was monitored in each well for about 30 h. Steady-state levels were reached in all three wells within about 2 h after the start of the tests. Table 1 gives the steady-state solution level and a local permeability estimate (from slug tests) for each well. The steady-state elevations in table 1 are evidence that a stable piezometric surface exists from 150 to 650 ft below the rubble surface, in the immediate vicinity of operating injection wells.

³Erskine, C. Hydrologic Drop Test In Situ Leach Operations. Magma Mining Co. internal rep., Mar. 1987, 5 pp.; for inf., contact C. Erskine, Pinto Valley Mining Co., Miami, AZ.

Table 1.—Drop test results in three wells

Well	Time to reach steady state, min	Steady-state elevation ft msl	Effective permeability ft/s	Saturated thickness, ¹ ft
38	110	2,590	26×10^{-7}	1,465
42	50	2,153	6×10^{-5}	1,028
45	130	2,575	7×10^{-6}	1,450

¹Above the undercut level.

²From slug test conducted in a nearby well.

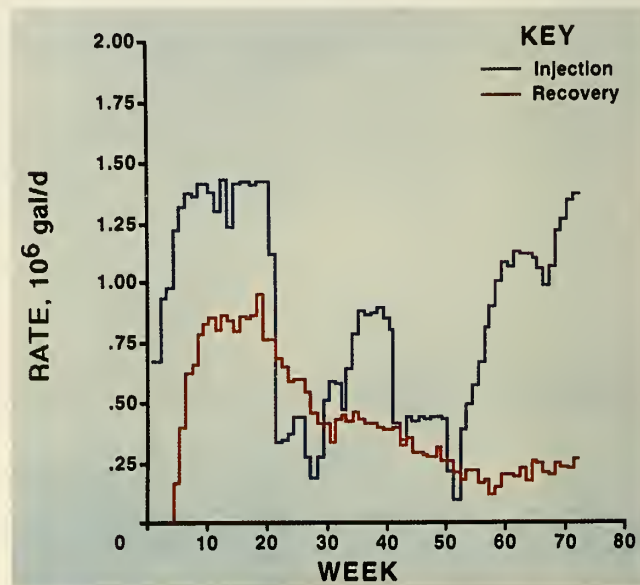


Figure 2.—Injection and recovery history through week 72.

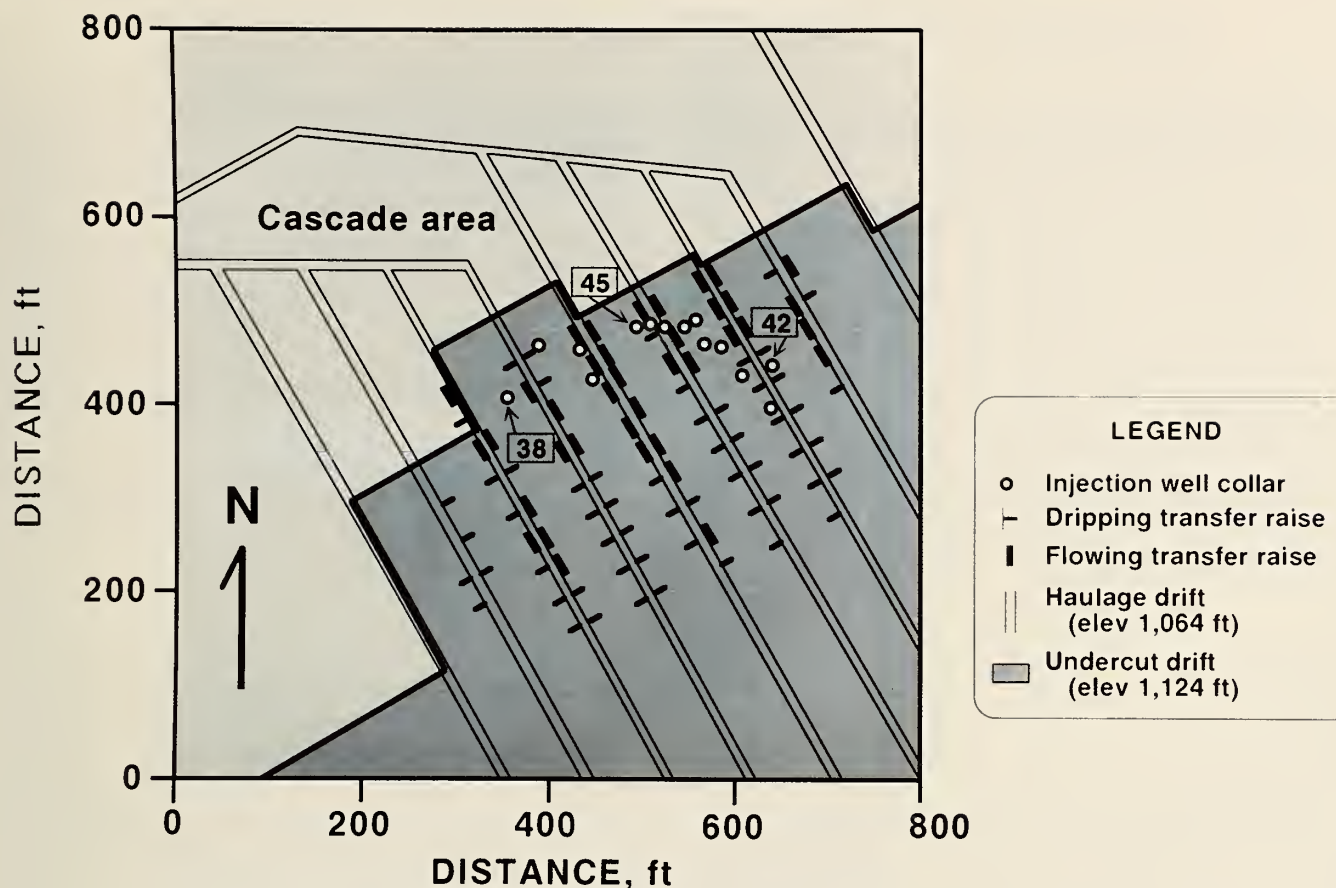


Figure 3.—Plan-view injection well locations and collection area.

MODELING MINE INFILTRATION USING DISTRIBUTED AREA SINKS

In two-dimensional flow problems, an area sink is a distribution of sinks over an area. Steady-state infiltration from a saturated aquifer into an overlying or underlying underground mine void can be modeled, in plan view, by one or more distributed area sinks, provided the size and shape of the area sinks can be chosen arbitrarily, to conform to the distribution of infiltration into the mine. In such a model, the rate of infiltration per unit area of mine workings, γ , is given by

$$\gamma = \frac{\phi^* - \phi(z)}{c}, \quad (1)$$

where $\phi^* - \phi(z)$ is the difference in piezometric elevation between the mine void and the overlying (or underlying) aquifer at a location z ; z is a complex variable used to represent a two-dimensional plan view coordinate location; and c is the resistance to vertical flow through a layer of saturated material located above (or below) the mine. By application of Darcy's law in equation 1,

$$c = \frac{h}{k}, \quad (2)$$

where h is the thickness of the resistance layer and k is its vertical permeability. The use of a resistance parameter in equation 1 to describe both the permeability and the thickness of a saturated layer implies that flow through the resistance layer is assumed to be one-dimensional.

A distributed area sink solution that satisfies equation 1 at every point z in an aquifer above (or below) a mine void can be obtained by integrating the solution for a point sink over an area. This integration is cumbersome, however, and may be carried out analytically for only a few, relatively simple shapes. Alternatively, a solution developed using the method of analytic elements can be used to approximate the distribution of piezometric head, $\phi(z)$, at every point z in the aquifer. The advantage of the analytic element methods is in the ease with which an approximate solution can be obtained for flow problems involving arbitrarily configured mine voids.

DISTRIBUTED AREA SINK SOLUTION USING ANALYTIC ELEMENTS

A full derivation of the distributed area sink solution using the method of analytic elements, together with a discussion of its use in representing infiltration sources such as ponds, rivers, and lakes, is given by Strack.⁴ For completeness, a brief outline of the method is presented here, with particular reference to conditions of underground mine infiltration.

Making use of the Dupuit-Forchheimer assumption, an area sink potential, $\Phi = \Phi(\phi, h, k)$, can be defined as a function of piezometric head, $\phi = \phi(z)$, saturated thickness, h , and permeability, k , that is continuous in both confined and unconfined mine inflow settings.

Considering a contour C enclosing a domain D^+ to be the arbitrary plan-view boundary of an underground mine void and using γ to denote a constant distribution of inflow to (or outflow from) the mine, then following Strack, an area sink solution for representing the mine void consists of determining an area sink potential, $\Phi = \Phi(z)$, at a location z , which fulfills the differential equation

$$\nabla^2 \Phi = \gamma \quad (3)$$

for any z in D^+ ,
and the differential equation

$$\nabla^2 \Phi = 0 \quad (4)$$

for any z in D^- ,
where D^- is outside of C .

The problem is solved by splitting the potential Φ into two parts:

$$\Phi = \Phi^i + \Phi^e, \quad (5)$$

where Φ^i is any function that satisfies $\nabla^2 \Phi^i = \gamma$ for z in D^+ , and where $\Phi^i = 0$ in D^- . By itself, Φ^i is discontinuous across the boundary C and therefore violates the conditions for continuity of both pressure and flow; Φ^e is used to eliminate these discontinuities. In order for the potential Φ (and therefore the pressure head ϕ) to be continuous on the boundary, Φ^e must jump by an amount

$$\Phi^{e-} - \Phi^{e+} = -\Phi^i \quad (6)$$

across the boundary, where the superscript (+) and (-) refer here to the value of Φ^e inside and outside of C , respectively. Additionally, if Q^i represents the component of discharge due to Φ^i that is normal to the mine boundary and Q^e represents the normal component due to Φ^e , then in order for flow to be continuous, Q^e also must jump by an amount

$$Q^{e+} - Q^{e-} = -Q^i \quad (7)$$

across the mine boundary.

A line integral that is a distribution of dipoles oriented normal to a line element is referred to as a "line doublet" element. The line doublet has the property that the potential Φ is

discontinuous across the element. Line doublets are placed along the boundary C in order to generate the necessary discontinuity in potential Φ^e . In a similar fashion, the necessary discontinuity in the normal discharge vector, Q^e , is produced by placing line sink elements (line integrals that are distributions of sinks) along the boundary C .

Thus the function Φ^e , which can also be interpreted as the real part of a complex potential $\Omega^e = \Phi^e + i\Psi^e$, may be written as a complex contour integral consisting of doublet and line sink elements. The function Ω^e is the complex potential that is valid at z locations outside the area sink; the potential valid at z locations inside the area sink is obtained by adding Φ^i to the real part of Ω^e .

The complex potential function Ω^e is given as

$$\begin{aligned} \Omega^e = & \frac{-1}{2\pi i} \oint_C \frac{\lambda(\delta)}{z - \delta} d\delta - \frac{1}{2\pi} \oint_C \frac{\mu(\delta)}{z - \delta} d\delta \\ & + \frac{Q_0}{2\pi} \ln(z - z_1), \end{aligned} \quad (8)$$

where the first term is the complex potential for a line doublet, the second term is the complex potential for a line dipole, and the second and third terms together constitute the potential for a line sink. The condition of continuity of Φ and Ψ along C is expressed in equation 8 in terms of density distributions for line doublets, λ , and line dipoles, μ , respectively. The total discharge from the area sink, which is not necessarily known in advance is Q_0 . The integration in equation 8 is performed analytically on an area sink boundary C that has been discretized as a polygon. The starting point of integration on the boundary is z_1 .

For a constant infiltration rate γ , the potential function Φ^i is given as

$$\Phi^i = 1/2 \gamma \xi^2, \quad (9)$$

where ξ is a unit length along the boundary of the area sink.

An area sink may be infiltration-rate specified or head specified. In cases where the area sink infiltration rate is known, the complex potential (Ω , where $\Omega = \Omega^i$ inside an area sink and $\Omega = \Omega^e$ outside an area sink) can be calculated directly. In cases where area sink head ϕ^* and resistance c are known, equation 1 is used to determine the infiltration rate per unit area γ .

The system of equations that results when multiple area sink solutions are superimposed, or when area sink solutions are superimposed on other analytic element solutions, such as wells, is solved directly in MINEFLO using Gauss elimination. In an unconfined flow setting, the aquifer thickness is itself the piezometric head; thus, the function relating potential, Φ , to piezometric head, ϕ , is nonlinear. In this case, the infiltration rate for an area sink that is head and resistance specified is calculated iteratively by MINEFLO.

⁴Work cited in footnote 2.

DROP TEST SIMULATION

Modeling the distribution of saturated conditions during weeks 26 to 41 of block-cave leaching operations consists, essentially, of superimposing 3 distributed area sink solutions and 15 steady-state well (point source) solutions in order to simulate the drop tests conducted in wells 38, 42, and 45. In the simulation, just as in the field test, 13 of 14 injection wells are specified on the basis of a known fixed head (2,850 ft msl), and a condition of zero injection is individually specified for each of wells 38, 42, and 45.

REPRESENTING THE DROP TEST SETTING

Figures 4 and 5 describe the setting of simulated drop tests in plan view and in cross section. The plan-view boundaries of the three area sinks used to represent collections of dripping and flowing transfer raises during drop tests are shown in

figure 4. The area sinks are at the undercut level (1,125 ft msl). Their boundaries are chosen so that each sink underlies one of the three wells (38, 42, and 45). Thus, the three area sinks are referred to by these numbers also.

Figure 5 is a schematic cross section showing components of an area sink used to simulate the well 45 drop test. The zone of saturated rubble material above the area sink in figure 5 is assumed to be composed of a resistance layer and a source aquifer. Flow within the resistance layer, which is a 716-ft thickness of rubble material located between the bottom of the injection well slotted interval (1,840 ft msl) and the undercut level (1,125 ft msl), is assumed to be downward only. The source aquifer in figure 5 is interpreted as a saturated zone located between the top of the resistance layer (1,840 ft msl) and the local piezometric surface (2,575 ft msl at well 45). Flow in the source aquifer is modeled as two-dimensional and unconfined.

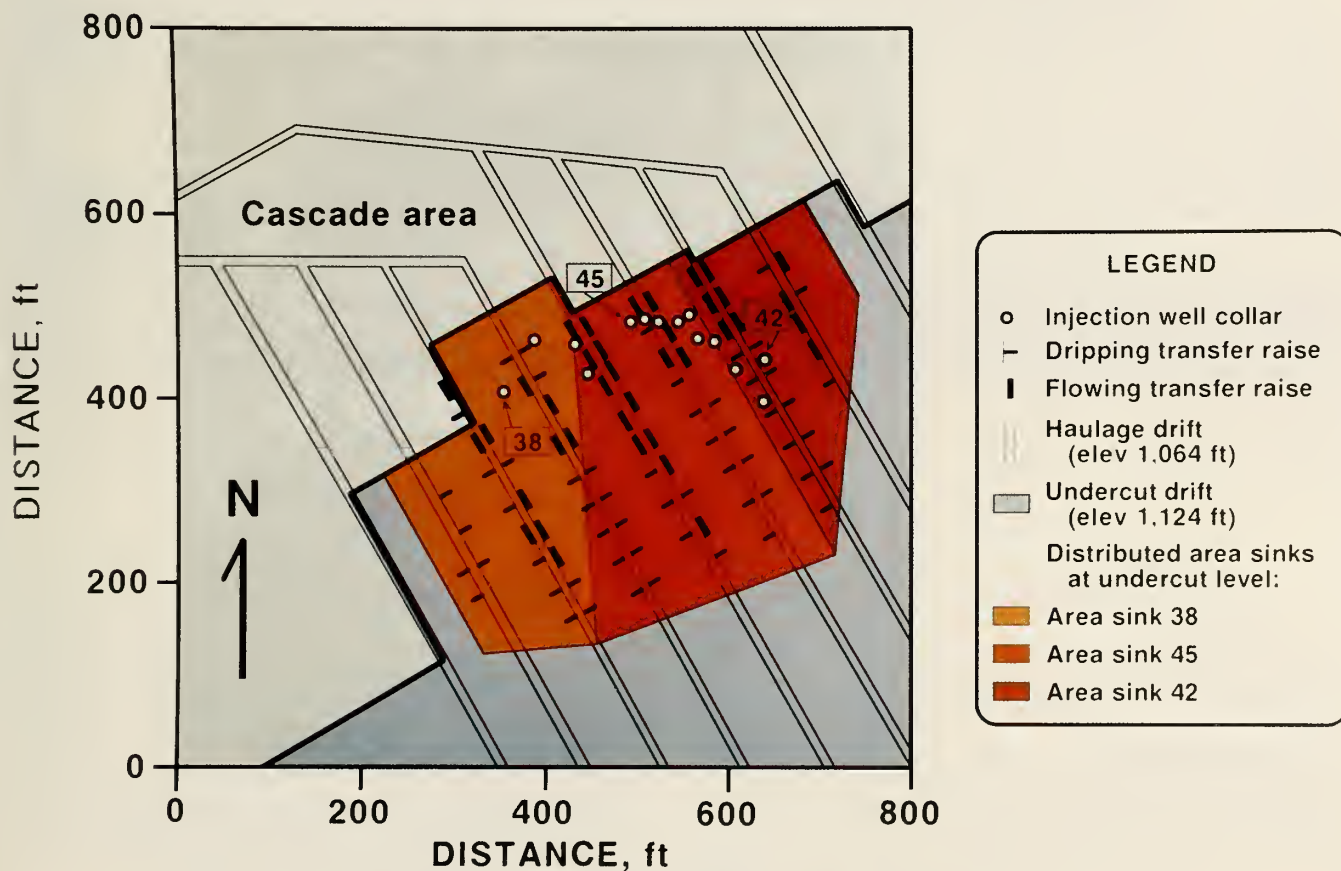


Figure 4.—Area sink representation of mine infiltration.

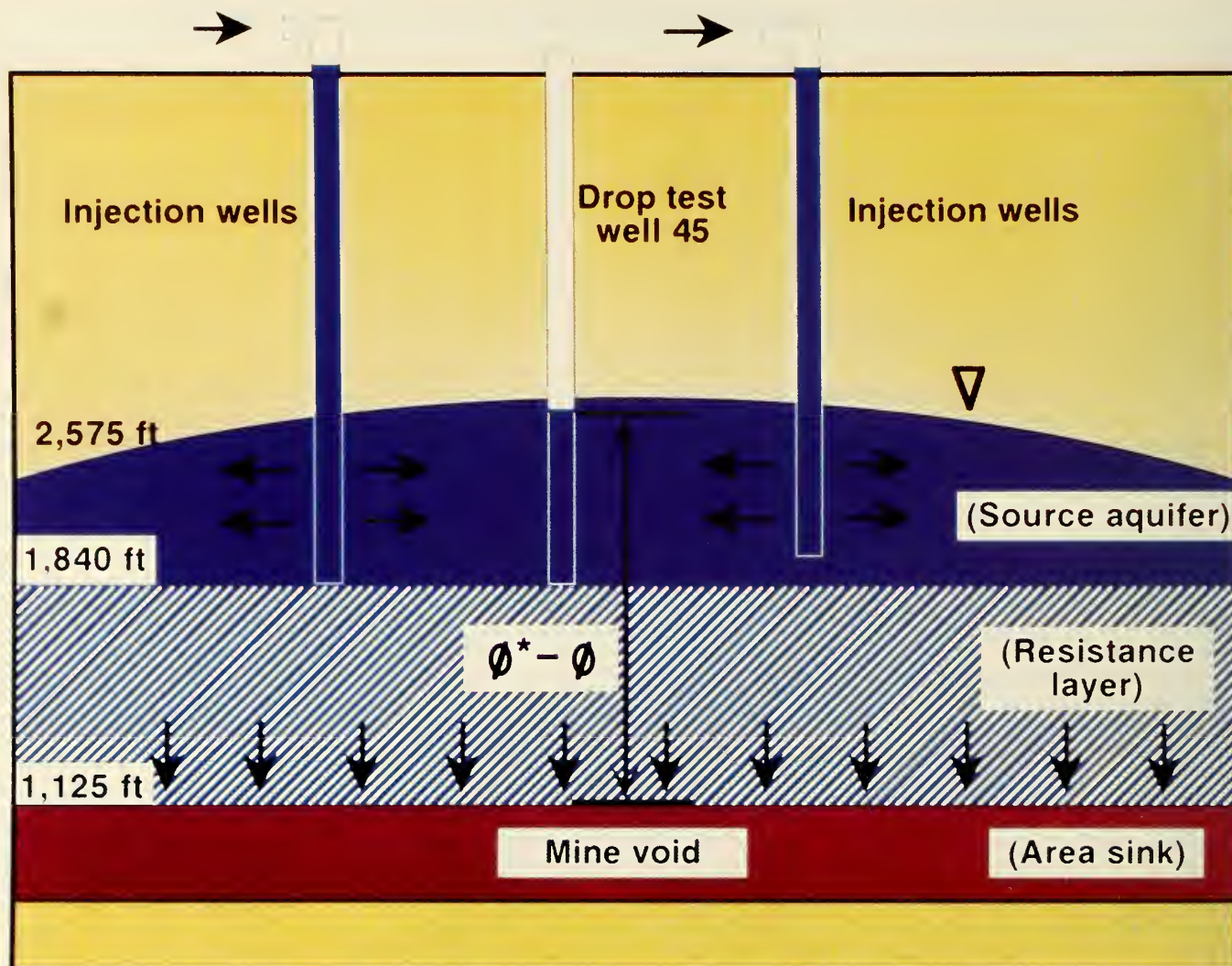


Figure 5.—Schematic cross section of distributed area sink.

AREA SINK CALIBRATION

Calibration consists of determining the infiltration rate for each area sink, such that the calculated heads at the locations of wells 38, 42, and 45 match the measured heads given in table 1.

Rearranging equation 1 to incorporate the definition of the resistance parameter c given in equation 2 and writing the infiltration rate per unit area γ as Q_c/A_c , where A_c is the plan-view area of a distributed area sink, gives

$$\phi^* - \phi(z) = \frac{Q_c}{A_c} \frac{h}{k} \quad (10)$$

In equation 10, estimates of resistance layer permeability, k , and thickness, h , are available from the drop test results in table 1 and from the definitions in figure 5. The plan view area A_c for each of the three area sinks is laid out in figure 4, and ϕ^* is simply the elevation of the undercut level. The difference in piezometric elevation, $\phi^* - \phi(z)$, between the source aquifer and the area sink is available at just one location z above each area sink, i.e., at the locations of wells 38, 42, and 45 (see table 1).

It follows that for area sinks that are head and resistance specified, a calibrated distribution of infiltration is one in which the three Q_c values chosen (one for each area sink) satisfy equation 10 at the locations of the three test wells. Because of the uncertainty associated with the permeability values obtained from slug tests, a limited amount of manipulation of resistance values c is permitted during MINEFLO calibration runs. By contrast, the value of $\phi^* - \phi(z)$ at the location of the test wells is known with much greater certainty. The calibration also takes advantage of the fact that the *total* well injection rate and the *total* mine outflow rate are known precisely during the drop tests.

DISTRIBUTION OF SATURATED CONDITIONS

The results of calibration are presented in tables 2 and 3. The actual and simulated $\phi^* - \phi(z)$ values at the locations, z , of wells 38, 42, and 45 are given in table 2. The discrepancy between the two sets of values is small, relative to the total thickness of saturated material, and thus equation 8 is considered to have been satisfied. In addition, table 2 shows the total well injection rate, actual and simulated. Table 3 gives

the proportional distribution of (total) mine infiltration, i.e., 367,500 gal/d, used to obtain these results. In the calibrated model, area sinks 42 and 45 account for about 95 pct of total solution recovery, while area sink 38 accounts for only 5 pct. The distribution is consistent with other estimates of plume size and shape⁵ and with the in-mine survey results of figure 3, which show a dense cluster of flowing transfer raises in the vicinity of wells 42 and 45 and mainly dripping transfer raises spread out in the vicinity of well 38.

In comparing tables 2 and 3, it is seen that a high infiltration rate per unit area of mine workings is associated with an overlying test well that has a low-steady-state piezometric head; compare, for instance, well 42 and area sink 42. Conversely, a low mine-infiltration rate is associated with a test well that has a high piezometric head; compare well 38 and area sink 38. This apparent contradiction between piezometric condition and mine infiltration is resolved when it is recalled that the operational injection wells are *head specified* in MINEFLO, not flow specified. Thus, given the same concentration of injection wells and the same injection well head specification (2,850 ft msl), the observation of a low head in a test well above the mine implies the existence of greater permeability in the vicinity of the well and therefore lower resistance to the downward flow of leach solution. The observation of a high head in a test well implies lower permeability and greater resistance to downward flow.

In the case of well and area sink 45, a well head that is comparable to well 38 is associated with an area sink infiltra-

tion rate that is about half that of area sink 42. The elevated head in well 45 is due to a greater concentration of head-specified injection wells (7 of 14 injection wells overlie area sink 45). As with all three area sinks, the infiltration rate per unit area of area sink 45 is proportional to the overlying permeability (see table 1).

The saturated plume that surrounds the injection wells in the rubble ore mass and in the cascade area can be described using the distribution of mine infiltration in table 3. The simulated plume is shown in figure 6. The plume boundary is a

Table 2.—Area sink calibration results, week 41

Condition	Simulated	Actual
Piezometric elevation above the undercut level, ft:		
Well 38	1,440	1,465
Well 42	1,004	1,028
Well 45	1,520	1,450
Total injection rate from all wells gal/d ..	876,000	875,000

Table 3.—Area sink outflow distribution

Area sink location ¹	Area, ft ²	Outflow rate, (gal/d)/ft ²	Total outflow rate, gal/d	Percent of total
38	54,137	0.36	19,438	5
42	21,872	7.72	168,849	46
45	49,912	3.59	179,216	49
Total simulated outflow			367,503	100

¹In reference to the overlying test well.

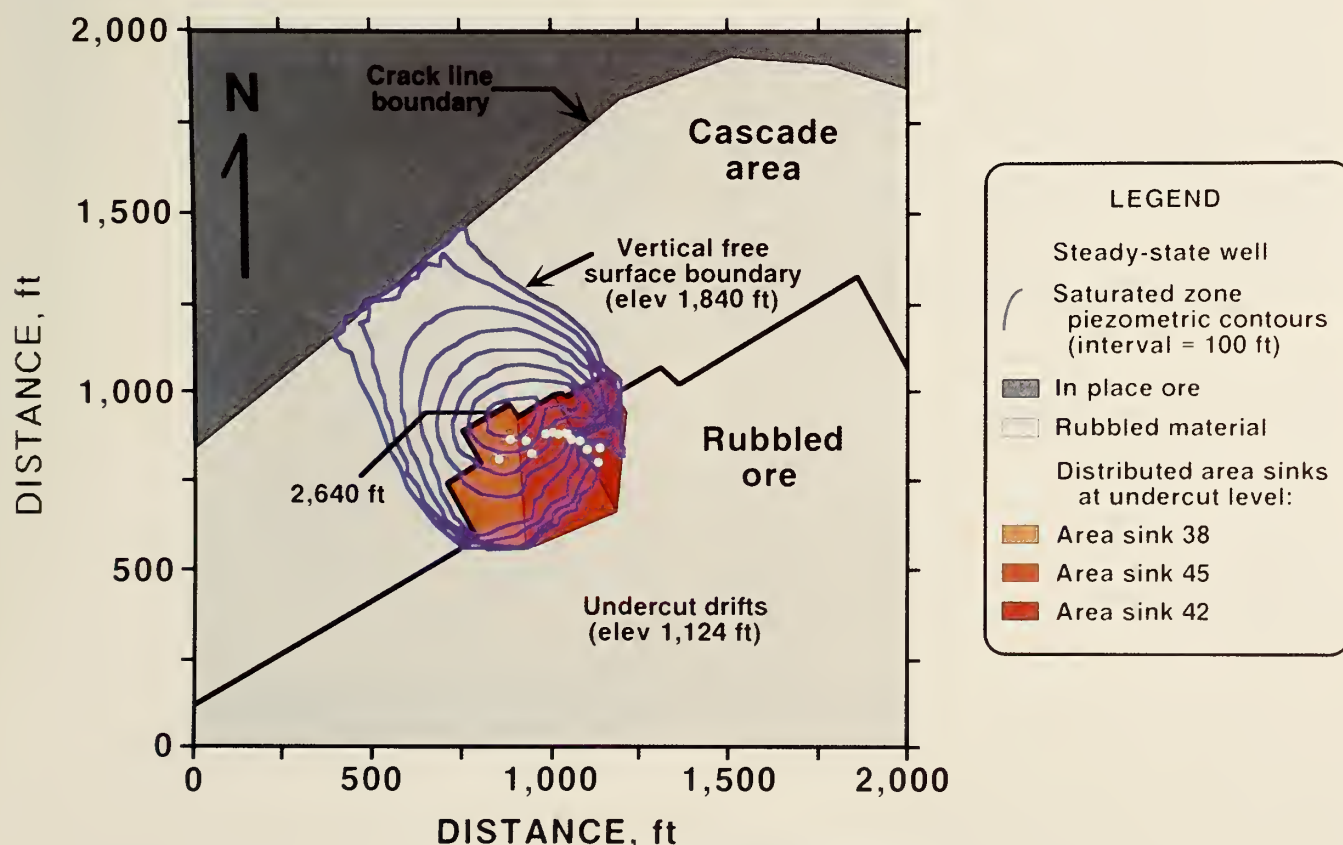


Figure 6.—Simulated solution plume, week 41.

⁵Swineford, G. Volume of Ore Wetted by In Situ Leaching. Magma Mining Co. internal rep., July 1987, 16 pp.; for inf., contact M. Miller, Magma Mining Co., San Manuel, AZ.

zero-head condition in the source aquifer; in the resistance layer, it is a vertical free surface extending from the bottom of the source aquifer (1,840 ft msl) to the undercut level (1,125 ft msl). It is not modeled as a zero flux condition, since figure 2 indicates that the plume boundary is expanding during almost

all of the first 41 weeks of leaching operations. The presence of a large portion of the plume in the cascade area results from the assumption that there is no permeable connection between the cascade area and the underlying haulage drifts.

PERMEABILITY ESTIMATION USING AREA SINKS

Consider the inverse of the previous calibration problem. The ore permeability is not known in advance, but the distribution of infiltration within the mine has been determined in an in-mine survey, which precisely measured the discharge from individual sump areas on the haulage level or possibly even from individual transfer raises. Rearranging terms in equation 10, the unknown permeability, k , can be written as

$$\frac{k}{h} = \frac{Q_c}{A_c(\phi^* - \phi(z))}. \quad (11)$$

Equation 11 is identical to the expression used for estimating aquifer recharge due to vertical leakage through a

deposit,⁶ where k/h (the inverse of resistance) is called the leakage coefficient, and A_c is the area over which vertical leakage is diverted to a pumping center.

If, in addition to the above in-mine survey, steady-state drop tests are conducted at well locations carefully chosen to provide representative measures of the head distribution in the overlying solution plume, then the MINEFLO computer program and the distributed area sink solution can be applied to determine a leakage coefficient k/h and a vertical permeability k , which satisfies equation 11. The method is appropriate for an ongoing block-cave leach setting in which it can be assumed that $[\phi^* - \phi(z)]/h$ measures the average vertical gradient in head above the area sink and that $\phi^* - \phi(z)$ and Q_c/A_c are constant with respect to time.

SUMMARY AND CONCLUSIONS

Distributed area sink solutions developed using the analytic element method have been shown to be useful for modeling irregular underground mine infiltration areas. Together with superimposed analytic element solutions for wells and other flow features, area sinks can be used to describe the saturated plume that develops above an underground mine collection area during block-cave leaching.

The area sink model aids in describing two important hydrologic variables that influence the size and shape of the saturated plume: the placement of injection wells above the underground solution-collection area and the occurrence of permeability layering in the rubbled material overlying the collection area.

The area sink and well solutions can be applied via an inverse problem to estimate the leakage coefficient and

permeability of saturated material overlying the in-mine collection area. For instance, the entire in-mine collection area can be represented by a single distributed area sink, a single test well can be used to estimate the average head in the saturated zone above the area sink, and then a single average leakage coefficient and permeability can be calculated for the entire saturated, rubbled zone. Alternatively, the in-mine collection area can be divided into several area sinks based on differences in infiltration rate. Steady-state head measurements can be obtained at one or more locations in the saturated material above each area sink, and the permeability of each portion can be estimated individually.

⁶Walton, W. C. Groundwater Resource Evaluation. McGraw-Hill, 1970, 664 pp.

HYDROLOGIC: AN INTELLIGENT INTERFACE FOR THE MINEFLO HYDROLOGY MODEL

By Michael E. Salovich¹

ABSTRACT

The U.S. Bureau of Mines has developed an expert system to provide assistance to researchers using the Bureau's hydrology computer program MINEFLO. This expert system, HydroLogic, is a multifaceted computer program that addresses the various ways in which MINEFLO may be used, including the design and analysis of in situ hydrologic operations. With tutorials, templates, and diagnostic questions, HydroLogic combines educational resources with diagnostic reasoning to create an environment supporting the MINEFLO hydrology program. This added support reduces the level of technical difficulty in using MINEFLO and thereby opens the model to a wider range of possible users. As a consequence, MINEFLO's effectiveness as a hydrologic tool for the mining industry is enhanced.

INTRODUCTION

The design and analysis of in situ systems require an understanding of the various factors that interact and define the hydrology of the system. This is a complicated task because, over time, hydrologic parameters, such as aquifer permeability, solution injection rates, well locations, and solution recovery rates, interact and change. An impermeable layer may form, the leachant solution may alter the permeability of the constituent ore deposit, or the occurrence of fracture fault lines may increase as subsidence occurs during block-cave leaching. These changes, in turn, call for a response in mining operations. Injection rates may have to be raised, wells may have to be repositioned and drilled deeper, the chemistry of the leachant solution may have to be remixed, or the in situ operation itself may have to be reevaluated and discontinued. In other words, this type of analysis of in situ systems can be approached as a study of parameters—hydrologic parameters that are identified and manipulated so that their interaction and effects are understood. In situ mining is a dynamic process. Understanding these factors and being able to anticipate their interaction is vital for efficient in situ operation. In light of this, the Bureau of Mines has developed two computer programs that work together in the analysis of these hydrologic factors: MINEFLO and HydroLogic. These programs have

been developed to assist the mining industry in the planning and development of in situ mining operations, as part of the Bureau's effort to increase minerals extraction efficiency.

MINEFLO is a hydrologic simulator that provides detailed and general analyses of hydrologic systems. It can provide information about aquifer permeability, well discharge and collection rates, leachant streamline direction and velocity, changing head pressures, and the effects of fractures and impermeable zones. Although this hydrologic simulator is robust and versatile, prior understanding of the hydrologic terms and methods it employs is required. With the goal of making MINEFLO a useful tool for the mining industry, the Bureau developed HydroLogic—a companion computer program designed to help researchers use MINEFLO.

HydroLogic, the focus of this paper, is a multifaceted computer program that addresses the various ways in which MINEFLO can be used. With diagnostic questions, tutorials, and templates, HydroLogic combines educational resources with diagnostic reasoning to create an "expert support environment" for the MINEFLO hydrology model. In other words, HydroLogic functions as an intelligent interface for MINEFLO. In the sense that expert systems are computer programs that attempt to emulate the thoughts and actions of an expert in a particular field of knowledge, HydroLogic is an expert system that provides expert guidance to those using MINEFLO.

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ENVIRONMENT

HydroLogic was developed in HyperTalk, the script language of HyperCard, for use on Apple Macintosh computers.² HyperCard provides an excellent environment for the implementation of HydroLogic because it offers "hypermedia" capabilities (i.e., interactive graphics combined with text) and the flexibility to be linked to other computer programs and data bases. HyperCard's advanced graphics tools enable the HydroLogic expert system to present different combinations of pictures, diagrams, and text with "cursor sensitive" areas for simple communication. Figure 1 shows the main menu of HydroLogic. Each rectangle in this picture is designed to be a "graphic button," sensitive to the cursor. When a button is selected, the system interprets it as a command and responds accordingly. With this method, users can navigate HydroLogic's menus efficiently by communicating commands on a graphic level.

HyperCard also has the ability to act as a central control point for accessing other computer files and applications. With its "open" command, it can initiate other computer pro-

grams, and with its "read" and "write" commands, it can access and manipulate any other computer files residing in memory. These commands allow HydroLogic to read and manipulate MINEFLO's data sets or to start the program itself. This creates a workstation environment that eliminates the awkwardness of having to switch back and forth between the consulting program and the intended application.

Unlike many computer programs, HydroLogic is not an isolated system; it provides a dynamic environment that can interact with any resource on the computer. This is important because HyperTalk's control structures are limited in creating systems that involve complex recursion or symbol manipulation. Future development of HydroLogic, then, may be extended to include additional artificial intelligence languages or systems, such as PROLOG or LISP.

Versions of the MINEFLO hydrology model exist on the MicroVax, Harris, and Macintosh computers. At this time, versions of the Hydrologic system operate only on the Macintosh II, Macintosh SE, and Macintosh Plus. Because of the size of the HyperCard code and the possibility of creating large data sets with MINEFLO, a hard disk is necessary for optimum performance.

²Reference to specific products does not imply endorsement by the Bureau of Mines.

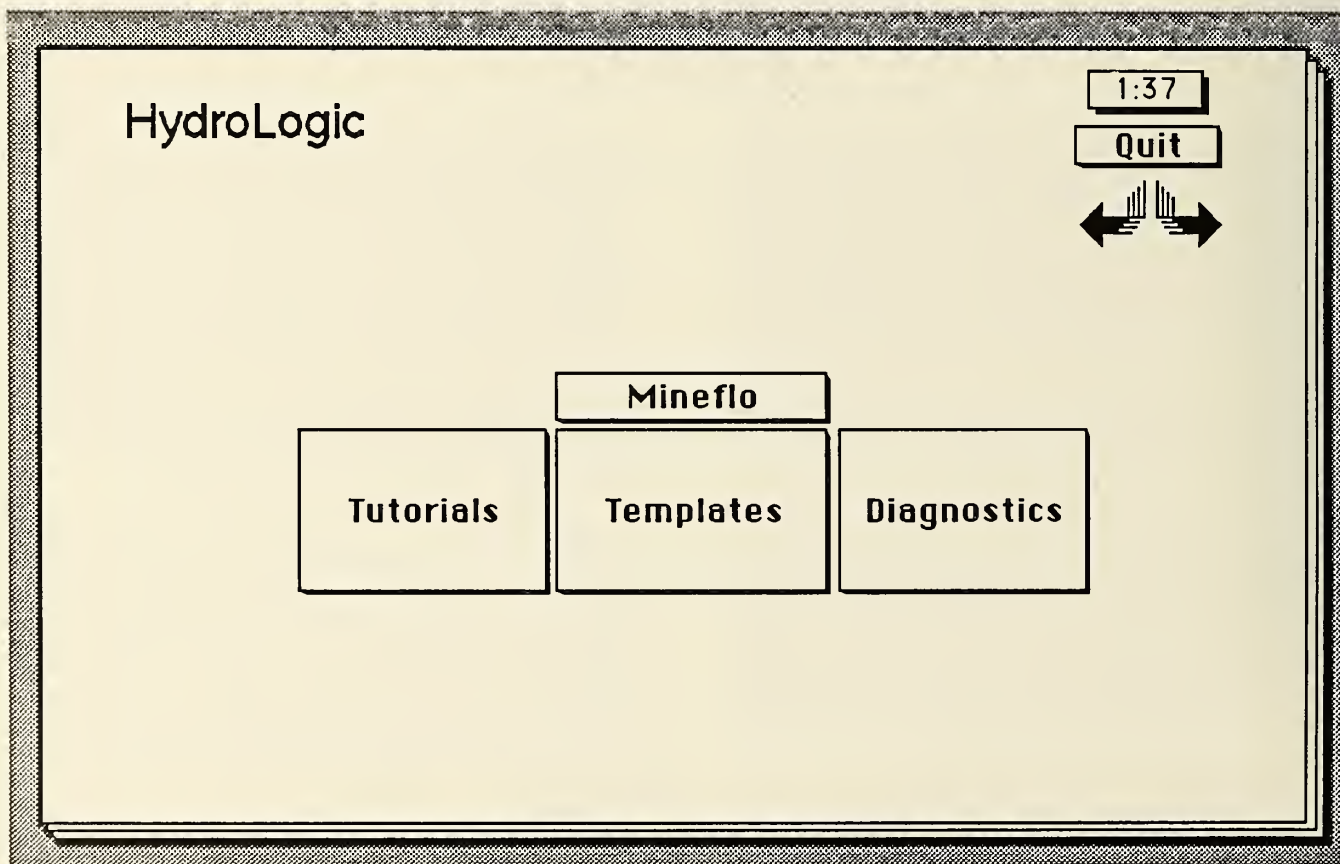


Figure 1.—HydroLogic's main menu, composed of graphic buttons.

DESIGN

The HydroLogic system is composed of three major sections: Diagnostics, Tutorials, and Templates. Only the Diagnostics section employs the diagnostic queries familiar in most expert systems. The Tutorials and Templates sections are educational resources that provide important information about the methodology and terminology used by the MINEFLO program. This combination of educational resources with diagnostic analysis enables HydroLogic to provide a broad base of support to those using the MINEFLO hydrology model.

DIAGNOSTICS

Most diagnostic systems require a small problem domain because of the way they capture and apply knowledge. This is done commonly through production rules—simple “IF . . . THEN” statements. These rules are carefully designed and structured so that they recreate the various steps of reasoning an expert may take in reaching a decision. Since all the mental steps involved in analyzing a problem must be explicitly represented in production rules, a problem must be sufficiently limited in its complexity to be addressed effectively by a diagnostic system. Otherwise, the diagnostic system would be too complex to design properly.

Additionally, diagnostic systems defined by production rules tend to be stiff and brittle. This is because production rules are predefined and their coherency can easily break down in the face of unforeseen situations. Since most of the production rules are interrelated, it is very difficult to make even the smallest changes to a system without a total reconfiguration. Therefore, the problem domain addressed by a diagnostic system should be sufficiently limited in complexity for reasons of effectiveness and efficiency.

The task of HydroLogic, as stated before, is to provide assistance to people using MINEFLO. The ideal expert, the model for the system, is someone who knows how to operate and apply MINEFLO in various types of situations. This, however, is a formidable task. The different combinations of MINEFLO input errors, user questions, and modeling problems are numerous and create a large problem space. This task is too large and complex to be accurately and efficiently represented by a single structure of production rules. However, subsets of this problem domain (i.e., specific topics concerning input parameters, error diagnosis of streamline generation, etc.) are less complex and therefore are viable subjects for smaller diagnostic systems. The large problem domain confronting HydroLogic could therefore be reduced to a collection of smaller and relatively simple subset units.

This translation of a large problem space into a collection of smaller units was the key for the development of HydroLogic. This method of “distributed problem spaces,” developed during the research, opened the way for creating expert systems for large or inexact problem domains. It is based upon the observation that a large or partially defined problem space can be mapped to a collection of smaller well-defined subsets. These smaller subset units are logically independent and highly specialized and therefore can be easily represented through production rules. Thus, an expert system, composed of these diagnostic subsets, can be created for a large or partially defined problem space.

HydroLogic, then, can be viewed as a collection of small diagnostic systems. These subsets are highly specialized, and some border on being trivial, but they are related under the common theme of being a resource for the MINEFLO

hydrology model. Figure 2 shows how the HydroLogic system enables users to select one of the diagnostic subtopics for consultation. Figure 3 shows a partial decision tree, depicting the typical scope and complexity of a diagnostic subtopic. These diagnostic subtopics use forward chaining to proceed from an initial (output problem) state to a goal (input revision) state. The initial problem state shown in figure 3 is the condition where the user of MINEFLO cannot start MINEFLO's flow streamlines. The rules in this case consist of all the required input specifications for generating streamlines using MINEFLO. The goal (revision) state is a condition in which the input error that prevented streamlines from starting is identified for the user.

If the logical structure formed by a set of production rules can be viewed as a decision tree, these subsets can be considered branches of a larger or, perhaps, partially defined tree. Therefore, the method of distributed problem spaces is a means of adding modularity to the design of a diagnostic system. An expert system, then, designed in this manner enjoys the same benefits as other modular programs: ease of design, maintenance, and modification. Small subsets are less complex and therefore are easier to create and manage. Together the subsets can create an extremely robust and flexible system. In addition, because the main problem is expressed in subtopics, users enjoy the benefit of having greater control over the system. One of the difficulties with expert systems is that they tend to interrogate users with a litany of seemingly trivial or inappropriate questions. With a modular program, users may select the topic of inquiry and thereby constrain the flow of questions.

As stated before, HydroLogic incorporates educational resources with the Diagnostics section. This is done because certain types of information are more easily and effectively communicated through references rather than through diagnostic analysis. For example, if a researcher needed to locate a menu of commands inside the MINEFLO program, it is apparent that viewing a structured diagram of MINEFLO's menu hierarchy would be more efficient than submitting to a diagnostic question-answer session. Therefore, for reasons of efficiency, the HydroLogic system was designed to include, in addition to its Diagnostics section, sections containing educational tutorials and templates.

TUTORIALS

HydroLogic's Tutorials section is a collection of interactive help files containing explanations and definitions to assist MINEFLO users with basic definitions and terminology associated with in situ hydrology. For example, tutorials have been developed to describe the analytic element methodology of the MINEFLO program. Specific tutorials exist for each of the analytic element solutions used in the program for modeling hydrologic features such as wells, permeability zones, barriers, fractures, etc. They contain information ranging from MINEFLO's mathematics to the various steps needed for the creation of a three-dimensional profile of an aquifer. The tutorials are interactive and easy to use, employing graphic buttons and other hypermedia features.

TEMPLATES

HydroLogic's Templates section contains well-defined examples of MINEFLO data sets. (MINEFLO uses data sets as a permanent means of recording and storing information about

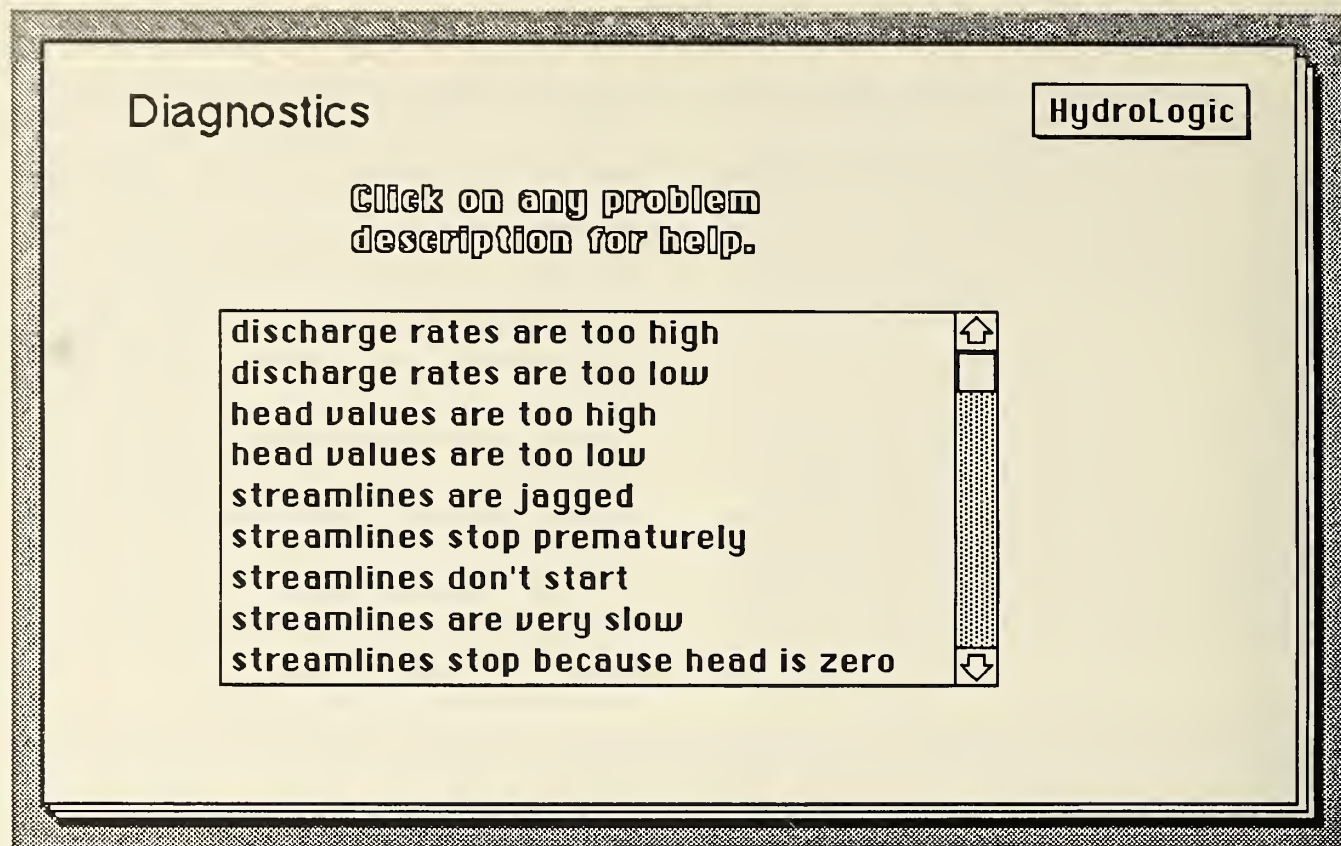


Figure 2.—HydroLogic's diagnostic subsets displayed in scroll box.

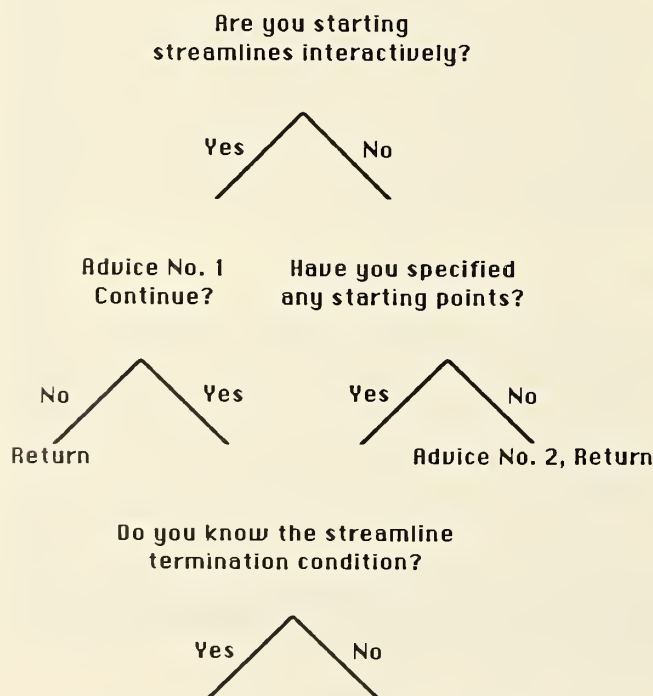


Figure 3.—Partial decision tree for diagnostic subtopic "streamlines don't start."

a particular hydrologic setting.) Each template is designed to convey important information about how to apply MINEFLO to a particular type of problem. Essentially, the templates are a way of imparting to MINEFLO users the Bureau's experience and expertise in identifying hydrologic concerns that are commonly associated with in situ mining activity. In this sense, they serve as starting points for researchers creating new data sets for the MINEFLO program. Users profit by running the MINEFLO program using these data sets and by manipulating the various input parameters. As researchers become more familiar with the program, they can build on the data sets and customize them, thus progressively generating their own insights and understandings about their particular hydrologic setting. In addition, the templates provide concrete examples of correct and acceptable input parameter definitions for the MINEFLO program.

Figure 4 shows an example of a template title card. Each card contains a title, a key of important features, and a graphic image generated by the MINEFLO hydrology model. The template shown in figure 4 is a fully defined data set used for modeling a single leachant injection well drilled above an underground mine void. It contains the basic hydrologic features characteristic of a method of in situ leaching called block-cave leaching. The key features of the data set are identified on the template, and users have the option of looking at additional notes, running the MINEFLO program with this data set, or copying and modifying it by adding more wells, changing the shape and location of the underlying mine, or adding other features.

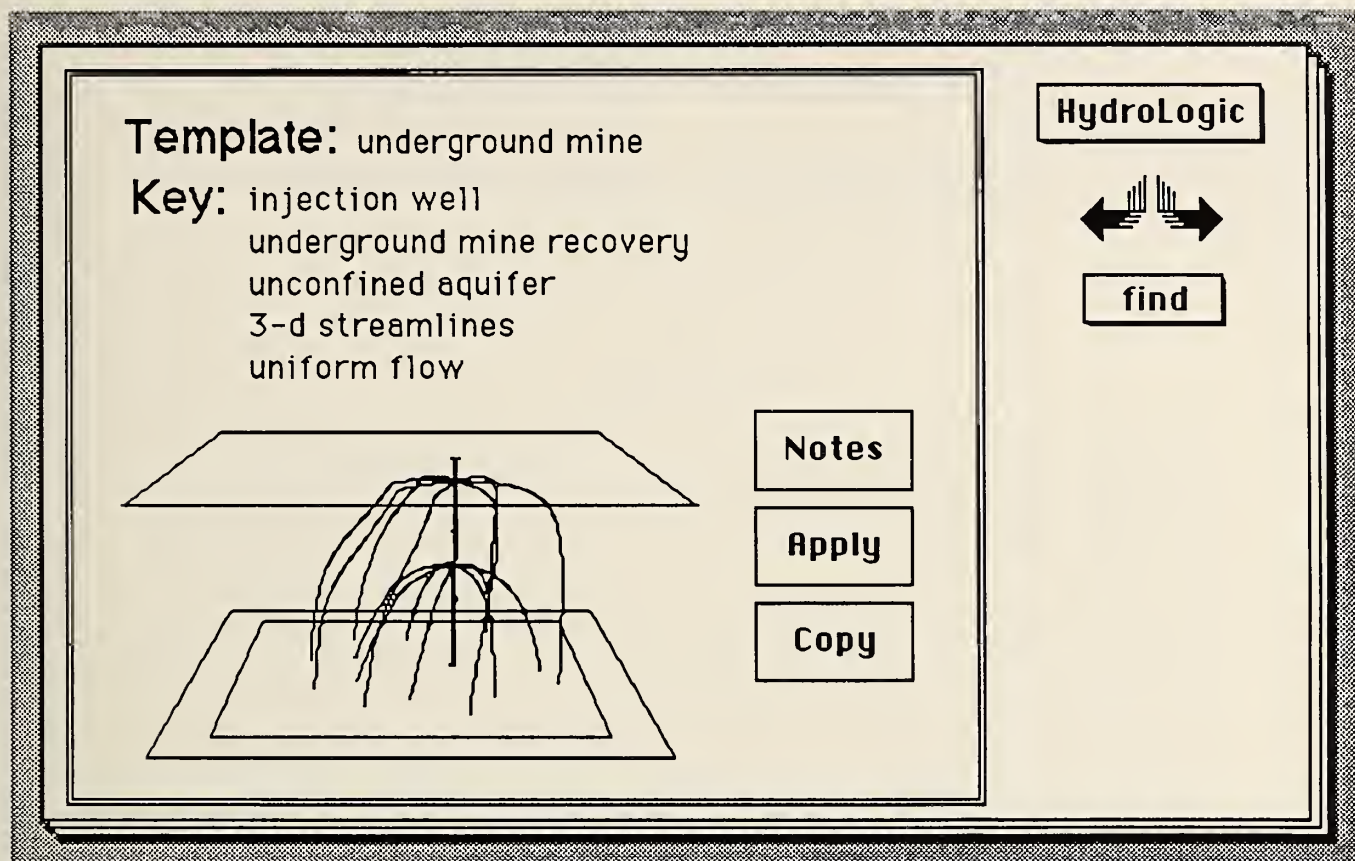


Figure 4.—Example of a template title card.

DISCUSSION

The primary purpose of HydroLogic and MINEFLO is to provide the mining industry with an effective tool for the design and analysis of in situ hydrologic systems. The MINEFLO hydrology model offers the capability to analyze the complex interaction of several types of hydrologic elements. HydroLogic functions as a companion program that provides information and support regarding MINEFLO's terminology and methodology. The measure of success of HydroLogic is its usefulness in providing support to MINEFLO. Although the basic steps have been taken in establishing diagnostic modules, tutorials, and templates for fundamental topics concerning MINEFLO, it is recognized

that further additions are necessary for the Hydrologic system to be considered comprehensive. As the Bureau's experience with the MINEFLO hydrology model grows, further development is planned for the HydroLogic system. It is also recognized that there are different types of computers used in the mining industry and that versions of MINEFLO and HydroLogic that operate on these computers should be considered. It is planned that, as other computer systems develop the necessary graphic and hypermedia capabilities required by these programs, versions of MINEFLO and HydroLogic will be developed for these systems.

SUMMARY AND CONCLUSIONS

In summary, HydroLogic contains three sections: Diagnostics, Tutorials, and Templates. Together they create a support environment for the MINEFLO hydrology computer program. Although they do not form a complete resource, they provide important information and perspective about MINEFLO's capabilities and operation. With the additional

support of HydroLogic, the task of learning how to operate MINEFLO is simplified. This added simplification, in turn, translates into greater productivity because it opens MINEFLO to a larger number of possible users and adds to the depth and quality of research performed with the model.

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PREDICTING AND MONITORING LEACH SOLUTION FLOW WITH GEOPHYSICAL TECHNIQUES

By Daryl R. Tweeton,¹ Calvin L. Cumerlato,² Jay C. Hanson,² and Harland L. Kuhlman³

ABSTRACT

The U.S. Bureau of Mines is conducting research to develop improved methods for predicting and monitoring the flow of leach solution during in situ mining. Potential benefits include more reliable assessment of leaching feasibility, higher metal recovery through better leach solution distribution, and greater confidence by regulatory agencies that leach solution will not escape from the mine.

The ability of seismic cross-hole tomography to detect fractured zones and saturated areas is being field tested for applications in predicting flow patterns and in monitoring leach solution injected above the water table. A tomographic analysis computer program, BOM-TOM, was written with special options for geophysical applications. It was first used with seismic refraction data for detecting fractured zones in a quarry. Tomographic analysis of seismic cross-hole travel-time data detected a mound in the water table made by injecting water between the source and receiver boreholes. Research to improve the reliability of tomographic reconstructions is continuing. Electromagnetic methods for determining where high-conductivity leach solution has replaced ground water are also being evaluated. Preliminary computer simulations indicate that surface-to-borehole time-domain electromagnetic induction is promising.

INTRODUCTION

IMPORTANCE OF PREDICTING AND MONITORING FLOW

Reliably predicting and monitoring the flow of leach solution is important in planning and operating an in situ mine. Predicting flow patterns is critical in assessing the feasibility of mining; if most of the flow is in a few large fractures, much of the ore may remain unleached. If mining is feasible, predicting and verifying flow patterns can aid in planning well spacings, selecting depths at which to perforate casings, and choosing pumping rates to provide the best leaching coverage of the ore. Monitoring the flow in a pilot-scale study can help in deciding if well spacing should be changed in a commercial operation. Better distribution of leach solution will result in its contacting more of the ore, which will increase recovery of the ore mineral.

Safeguarding the ground water resources near an in situ leach mining operation is a responsibility shared by the operator and regulatory agencies. Obtaining permits from regulatory agencies is one of the critical steps in starting an in situ mine. Improved methods of predicting and monitoring

flow could help mining companies control leaching solutions and assure regulatory agencies that these solutions will not escape from the well field.

ADVANTAGES OF GEOPHYSICAL METHODS

Bureau of Mines researchers believe that certain cross-hole or surface-to-borehole geophysical techniques could provide important advantages over present methods of predicting and monitoring flow. The present method for finding probable flow paths, such as fractured zones, is a combination of borehole geophysics and examination of cores. The volume investigated is limited to the region close to the boreholes. Important geologic features could be missed unless the boreholes are closely spaced. The present method of monitoring is to sample monitor wells. However, the flow of leach solution through an ore body is not uniform, especially in fractured hard-rock deposits, such as copper and most precious-metal ores. The inhomogeneous flow pattern can decrease the reliability of monitor wells for locating leach solution. Placing monitor wells sufficiently close together so that regulatory agencies are confident that leach solution cannot escape undetected can be expensive. At the 600-m depth at which in situ copper mining can occur, a monitor well costs several tens of thousands of dollars.

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Cross-hole geophysical methods with tomographic data analysis and surface-to-borehole methods can examine the region between boreholes. Thus, an important geologic feature is less likely to be missed. Used before injection, these methods could indicate probable flow paths. Comparing measurements made before and after injection could locate leach solution. An additional economic advantage of using geophysical systems over coring and sampling monitor wells is that the systems can be reused, which decreases the overall cost over the life of the mine.

PRIOR RESEARCH

Bureau contract research in 1979 (1)⁴ indicated that neither surface four-terminal resistivity nor controlled-source audio-magnetotellurics (CSAMT) reliably detected carbonate leach solution at a depth of 80 m at an in situ uranium mine in Wyoming. The surface resistivity measurements were hampered by the thinness of the leached zone compared with its depth from the surface. The CSAMT system suffered from interference, such as that from powerlines. However, technology has improved since that time, so that failure of the surface resistivity and CSAMT systems does not necessarily mean that those techniques would fail with modern geophysical equipment.

Tomography (explained in the section "Seismic Cross-Hole Tomography") has been applied to geophysical investigations by a number of researchers. Ramirez (2) reported the application of tomography to electromagnetic (5- to 40-MHz) at-

tenuation cross-hole data with borehole separations up to 30 m. Dines (3) analyzed electromagnetic (50-MHz) cross-hole data, measured both attenuation and velocity, and found attenuation to be more diagnostic. Applications to seismic reflection data were made by Chiu (4) and Bishop (5). Analyses of seismic cross-hole data were performed by Albright (6), Gustavsson (7), Peterson (8), and Peterson (9). Foss (10) applied tomography to detecting coal mine hazards with ground-penetrating radar (GPR) but reported that tomography did not conclusively show the clay vein of interest. Werniuk (11) described research being coordinated by Paul Young of Queen's University, applying tomographic imaging to mining-induced seismicity and rock burst phenomena. A helpful summary of the advantages and disadvantages of the various tomographic mathematical methods was given by Worthington (12).

RECENT AND CURRENT BUREAU RESEARCH

The Bureau's current emphasis is on seismic cross-hole tomography to predict and monitor the flow of leach solution. Preliminary experiments have been conducted to test the ability of this method to locate a curved water surface, simulating locating leach solution injected above the water table. In addition, seismic tomography has been used for detecting fractured zones in a quarry and in building stones. The Bureau is also evaluating electromagnetic methods to monitor leach solutions, which are described in more detail in a subsequent section.

SEISMIC CROSS-HOLE TOMOGRAPHY

BACKGROUND

The seismic cross-hole method consists of transmitting seismic waves from one borehole to another and measuring the corresponding velocity and/or attenuation. Measuring velocities is easier and the interpretation is usually easier. The seismic velocity is lower and the attenuation is greater in zones of fractures and broken rock than in solid rock. The seismic compressional wave velocity increases with saturation. In Bureau laboratory experiments using competent cores from various rock types, the velocity was typically about 10 pct greater in saturated rock than in dry rock (13). Bureau researchers expect that there will not be a measurable difference between the velocity in rock saturated with leach solution and that in rock saturated with ground water.

The general procedures involved in tomography have been described elsewhere (3, 8-9, 12, 14-15). Only a brief discussion, as it relates to this special application, will be given. Tomographic reconstruction as applied to cross-hole data is a mathematical process for constructing a two-dimensional representation of a field, such as seismic velocity between two boreholes. Figure 1 shows a simplified diagram of the source (transmitter) and receiver locations. In practice, there would be many source and receiver locations, so the region between the boreholes would be crossed by many ray paths. One method of collecting tomographic data consists of setting the source at one position, stepping the receiver through its positions, then moving the source to its next position and repeating the series of receiver positions. This pro-

cedure is repeated until all of the desired ray paths have been generated. Alternatively, the receiver position can be fixed, while the source position is varied. If the borehole is filled with fluid so that coupling is adequate without clamping the source to the borehole wall, the source can be moved continuously and fired at timed intervals.

Tomographic analysis of cross-hole seismic travel times provides a two-dimensional picture of the distribution of seismic velocities between the source and receiver boreholes. The positions of zones of fractures and leach solution can then be inferred from the seismic velocities. Albright (6) showed that seismic cross-hole tomography can indicate geologic structure, including zones of fractures. Research is being conducted to determine the reliability of using the method to show the location of leach solution. The method appears promising for detecting leach solution when the ore is unsaturated before injection. Thus, the seismic method may be applicable for detecting likely flow paths above or below the water table and for locating leach solution injected above the water table.

Compared with surface techniques, seismic cross-hole methods offer higher resolution, higher sensitivity to features in the area of interest, and lower sensitivity to features outside the area of interest. However, boreholes are expensive; to be practical, the cross-hole methods must use boreholes that are needed for other purposes, such as injection or monitor wells.

Compared with other cross-hole techniques, the seismic method has several advantages. Seismic waves can easily penetrate several hundred meters and propagate through casings. The equipment is relatively inexpensive. For example, an air gun source costs about \$12,000. A disadvantage is that this method cannot distinguish between ground water and leach solution.

⁴Italic numbers in parentheses refer to items in the list of references at the end of this paper.



Figure 1.—Source (transmitter) and receiver locations in boreholes.

TOMOGRAPHIC ANALYSIS COMPUTER PROGRAM

Bureau researchers wrote the tomographic analysis computer program BOMTOM (Bureau of Mines tomography)⁵ because available computer programs did not contain options needed for the most effective analysis of cross-hole data. To perform tomographic analysis, the region between the boreholes is mathematically divided into small cells called pixels. The tomographic reconstruction assigns a velocity to each pixel to provide the best fit to the data. Tomographic analysis of cross-hole data without constraints does not yield a unique reconstruction, even when the number of measurements exceeds the number of pixels. Thus, a unique reconstruction cannot be obtained by increasing the number of cross-hole measurements. Cross-hole data can be fit equally well by many different reconstructions, so appropriate mathematical constraints must be applied to help select the correct one. A tomographic program providing a test for nonuniqueness and constraints to counteract nonuniqueness was needed. Available programs did not provide such a test or constraints appropriate for in situ mining applications.

BOMTOM provides a test for uniqueness and constraints to help obtain unique reconstructions. It is easy to run on personal computers commonly used by mining and geophysical companies. Options can be selected interactively. BOMTOM uses the simultaneous iterative reconstruction technique (SIRT) with straight-ray paths. The adequacy of straight-ray paths for in situ mining applications will be examined in future research. A curved-ray program with suitable constraints is being developed for use in situations in which velocity contrasts are large and bending of rays is significant.

BOMTOM allows the user to test uniqueness by varying the pattern of the initial velocity guesses used to start the iterative solution procedure. A unique reconstruction is independent of the initial velocity guesses. BOMTOM provides the following options for constraints:

1. Maximum and minimum calculated velocities;
2. Known, fixed velocities in boreholes, as from sonic logs;
3. Horizontal layers near the top and/or bottom of the investigated area, in which seismic velocity does not vary with horizontal position;
4. Smoothing, in which the calculated velocity in a pixel is influenced by the velocities in neighboring pixels.

DETECTING FRACTURES WITH SEISMIC TOMOGRAPHY

DETECTING FRACTURED ZONES IN A QUARRY

BOMTOM was used for detecting fractured zones in the Rockwell lime quarry near Rockwood, WI, by analyzing seismic refraction data (17). The general principles behind detecting fractures in quarries and detecting fractures at an in situ mine are identical; namely, the seismic velocity decreases in fractured zones (18). Thus, examining the application of tomography to seismic refraction data can indicate its promise for analyzing seismic cross-hole data to detect fractured zones in in situ mines.

A Bureau report (16) describes BOMTOM and results of testing the effectiveness of various combinations of constraints in obtaining unique reconstructions when simulating in situ mining. Calculations were performed with synthetic data of the type expected from in situ mining above the water table, with a dip in the leach solution level between injection wells. The seismic velocity distribution was modeled by attributing a velocity increase of 10 pct to the leach solution. The simulations generated a high-velocity artifact near the dip between the wells. (A tomographic artifact is an error in the calculated velocity for the corresponding pixels.) The artifact was reduced by the constraint that velocity was independent of horizontal position near the top and bottom of the pixel grid. This can be a realistic constraint when the ore at the top of the grid is dry and the ore at the bottom is saturated, and there are no other factors (such as nonhorizontal bedding) causing a significant change with horizontal position near the top and bottom of the grid. Applying the additional constraint of fixing the velocities in the boreholes at known values reduced the artifact slightly more. The reconstruction improved significantly when the upper velocity limit was close to the highest velocity in the model. However, the appropriate upper limit may not be well known for field data. The simulations demonstrated the need to consider nonuniqueness when performing such calculations and the desirability of surrounding the investigated region by an area of known or constrained velocities. One way to impose constraints at the sides of the region is to measure the seismic velocities at intervals in the boreholes with sonic logs and fix the corresponding velocities to those values during the tomographic data fitting.

EQUIPMENT DEVELOPMENT

The Bureau developed an improved well-locking geophone receiver system. It provides a better signal-to-noise ratio than commercial seismic receiver units that the Bureau has leased. The improved signal-to-noise ratio allows a lower strength source to be used, thereby reducing the risk of damaging the well casing. It is smaller, easier to use, and less expensive than most commercial units. It will be described in a later publication after clarification of patent status.

A truck-mounted field system is being prepared for recording field data. It includes a generator, an air compressor, winches, and all equipment needed to make field arrangements. Field tests will be conducted in the type of rock and at the depths typical of in situ copper mining.

Bureau researchers needed a fracture detection technique to evaluate the success of blast design changes in reducing pit-wall damage. Standard seismic methods were inadequate, so seismic refraction tomography was employed. Each survey consisted of a series of refraction fan shots in which a line of geophones was arranged parallel to and offset from a line of shot points (fig. 2). Refraction tomographic surveys consisted of 24 shot points and 24 geophone stations, yielding 576 ray paths per survey. Seismic travel time data were used for imaging the seismic velocity distribution on the refracting rock layer.

Tomographic reconstructions were performed with BOMTOM. Travel times were accurate to ± 0.05 ms. Velocity constraints consisted of high and low limits and the use of a uniform initial velocity guess. Other optional parameters included the use of a delay time to correct for seismic wave travel

⁵Source and executable codes in FORTRAN on 5¼- and 3½-in IBM-compatible diskettes are available upon request from D. R. Tweeton, Twin Cities Research Center, Bureau of Mines, Minneapolis, MN 55417.

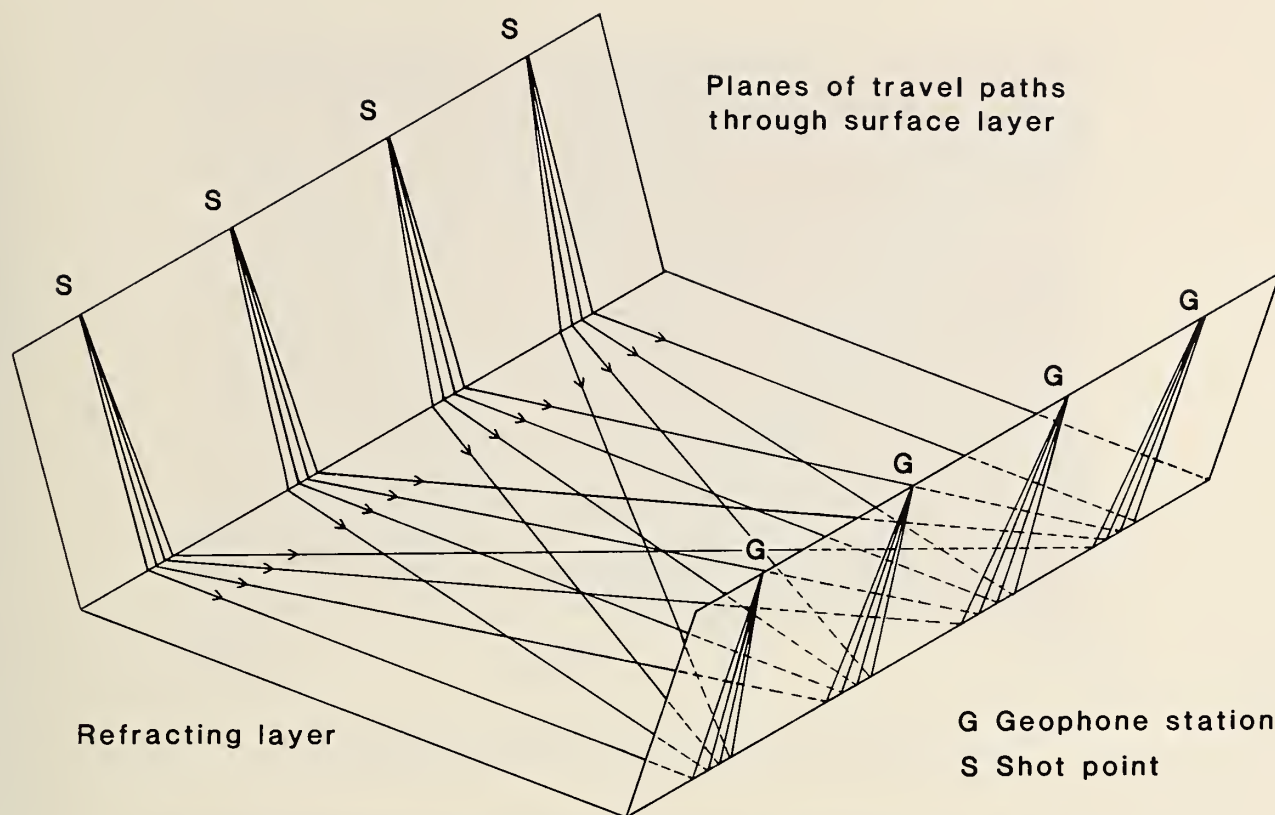


Figure 2.—Conceptual drawing of ray-path geometry for series of four refraction fan shots.

time through the surface layer and the use of a weighted smoothing routine after BOMTOM arrived at a final solution.

A production blast design was modified so that half of the last row of shot holes were loaded with the quarry's standard explosive load, while the remaining holes were loaded with a reduced-diameter explosive column in the stemming zone (fig. 3). This was done to reduce cratering while promoting shearing of the rock.

The results of the preblast and postblast surveys are shown in figures 3 and 4, respectively. The size of each pixel is 0.5 m. The most obvious feature in the preblast tomogram is a low-velocity trend from the lower right to near the upper left corner. Since a predominant joint set was found in the quarry with an average strike of N 26° W, it appears reasonable that this feature is the seismic velocity signature of a similar joint set intersecting the shallow refracting layer. The low-velocity zone in the upper right corner may be the expression of another joint set or it may be a zone of fracturing from a previous blast.

Comparing the preblast and postblast tomograms (fig. 5) shows that the velocity field of the refracting layer has changed. In the upper left and upper center areas, closest to the standard blast design, zones of relatively high velocity before the blast now appear as low-velocity zones. The low-velocity feature interpreted as jointing in the preblast tomogram is also more pronounced, especially in the lower right area. In addition, the high-velocity area nearest the shotholes with the modified blast design has remained about the same.

It appears that the changes made in the blast design in the five right-hand shotholes have contributed to a reduction of fracturing in the rock left standing, while the rock nearest the

shotholes with the standard load has been extensively damaged. It is possible, however, that much of the energy produced by the entire blast propagated into or was channeled through the preexisting jointed area, causing some additional fracturing, which makes this zone more pronounced in the postblast tomogram. In either case, the results of the seismic refraction tomographic analysis are consistent with known features of the rock.

There are two possible sources of error that users of this method should consider. One source of error is the uniform delay time for all ray paths, since lateral velocity variations were present in the surface layer as a result of the blast under investigation and subdrilling from the previous bench. The other source of error is the use of straight-ray paths. The velocity contrast in quarries is large enough to make curved-ray analysis desirable. Work is in progress to develop a curved-ray program with BOMTOM's features.

The methods applied during this study were successful in identifying major velocity trends in shallow refracting rock layers. The trends can be related with confidence to jointing and zones of intense fracturing caused by mine blasting. Furthermore, refraction tomography may hold great potential for future widespread application as a powerful and economic tool for mine planning and blasting design.

DETECTING FRACTURES IN BUILDING STONES

BOMTOM has been used by a private company for examining building stones by analyzing seismic travel times for two pairs of faces. Fractured zones that were not visible from the surface were detected with seismic tomography.

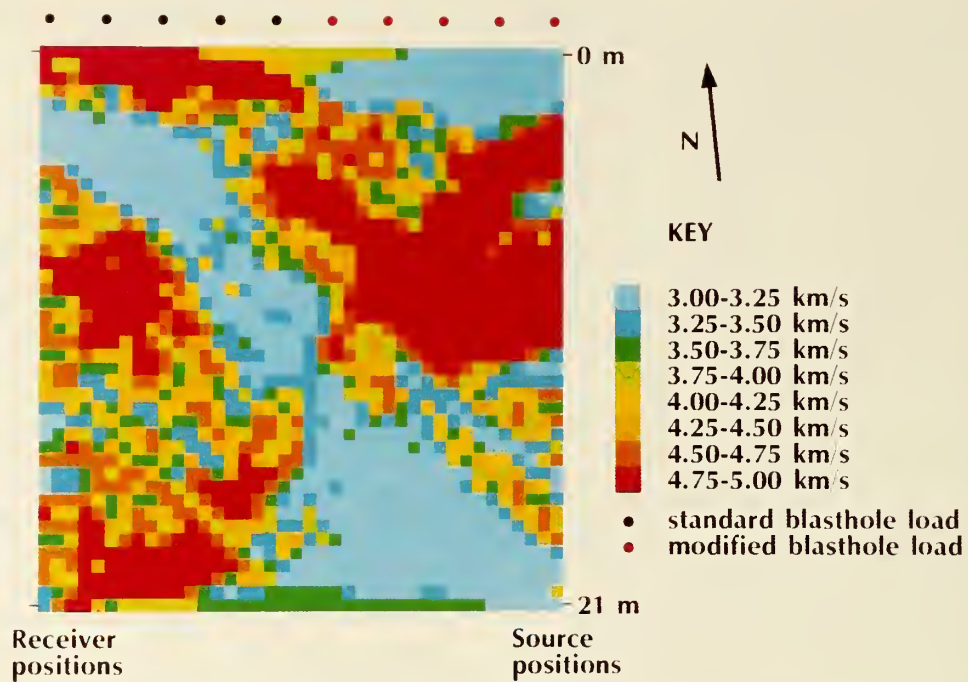


Figure 3.—Preblast tomogram and location of blast design variations.

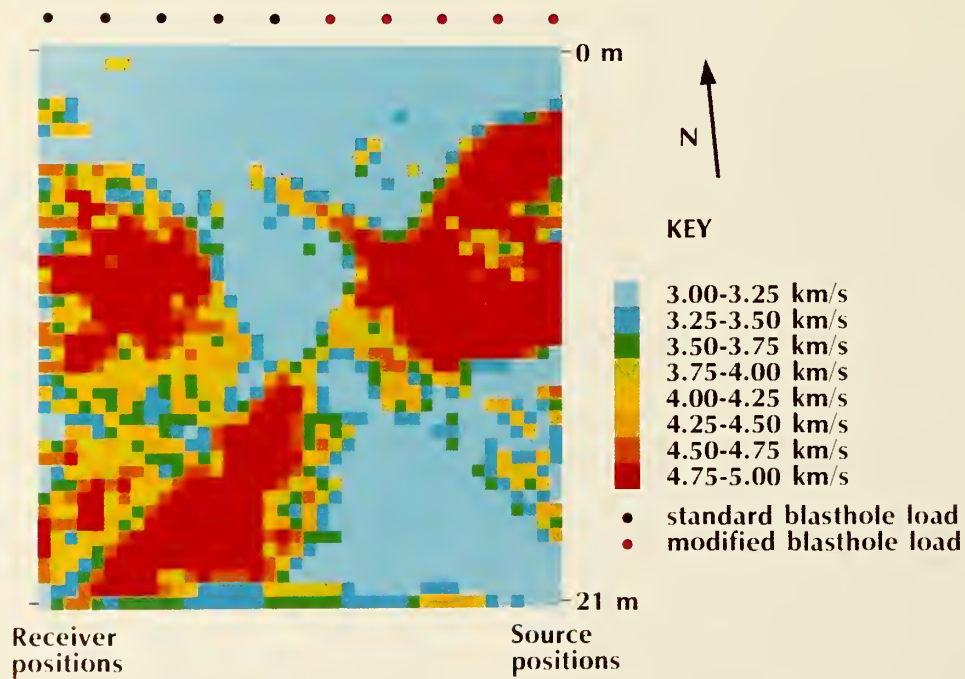


Figure 4.—Postblast tomogram.

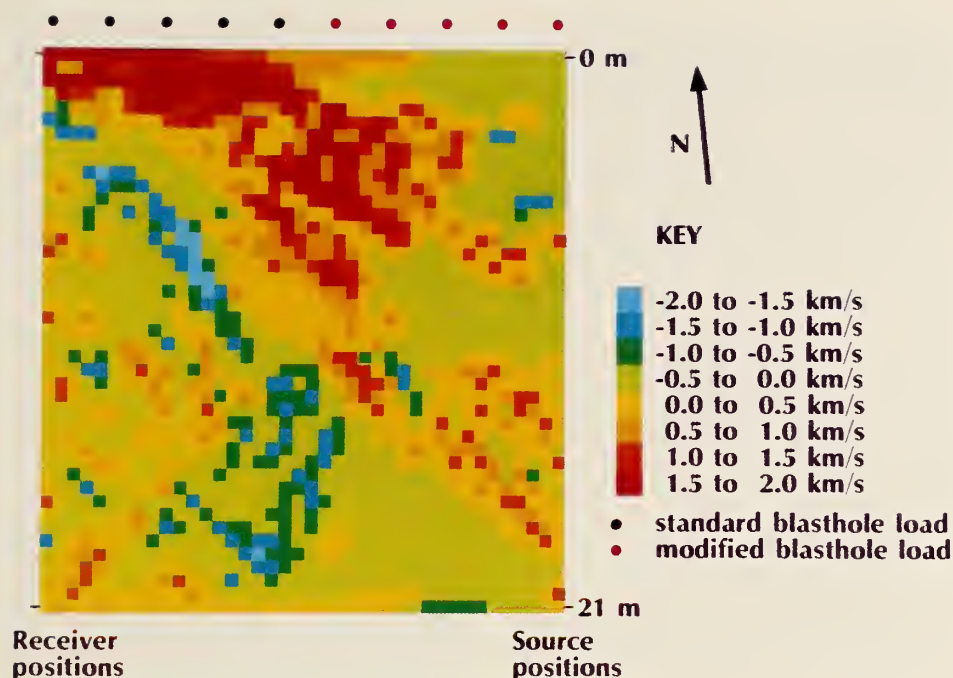


Figure 5.—Preblast velocities minus postblast velocities.

LOCATING CURVED WATER SURFACE WITH SEISMIC TOMOGRAPHY

PRELIMINARY TESTS AT BUREAU'S SITE

Preliminary experiments to evaluate the use of seismic travel times to locate leach solution injected above the water table were conducted at the Bureau's on-site geophysical test facility. Water was injected instead of leach solution because the change in travel times depends on physically wetting the rock rather than on the chemical properties of the solution. The experiment was conducted to help answer two questions:

1. Does the increase in the velocity of seismic waves in saturated rock observed in the laboratory (13) require the thorough drying and saturating used in those experiments, or will it also occur under field conditions?

2. If travel times do decrease as the rock becomes saturated, how well can tomographic analysis locate the regions where the rock changed from dry to wet?

Tests were conducted by transmitting a seismic signal between boreholes 4.2 m apart in fractured limestone. A mound in the water table was created by injecting water into a borehole between the source and receiver boreholes. The seismic source was an air gun. The receiver was a wall-locking triaxial borehole geophone. The source and receiver positions were 3.7 to 9.1 m below the surface, in steps of 0.3 m.

Data were first collected when no water was being injected. The water level in all boreholes was 6.1 m below the surface. After a tomographic data set was collected, water was injected into the middle well at about 40 L/min. This injection raised the water levels in the source, injection, and receiver

wells to 4.9, 4.2, and 5.3 m, respectively. A curved water surface was desired to test the ability of the tomographic reconstruction to show the shape of the top of the water surface between the source and receiver wells. A problem prevented collection of a complete tomographic data set. The air gun repeatedly became stuck in the uncased limestone borehole, so about one-fourth of the desired ray paths near the bottom of the grid could not be used.

The data⁶ were analyzed with BOMTOM, using smoothing. The reconstruction using preinjection data showed high-velocity zones corresponding to water at the expected level, but there were also high-velocity zones above the water. The reconstruction using data taken during injection showed the mound in the water where it should be, but mathematical artifacts and effects not related to the mound were present. There were at least two problems interfering with the tomographic analysis. First, the coverage was not complete. Second, the maximum calculated velocity was three times as great as the minimum calculated velocity, so the use of straight rays could produce artifacts in the reconstruction. Considering those problems, the presence of the mound in the reconstruction was encouraging. However, the results demonstrated the need to obtain data from a site that would allow more complete coverage.

⁶Data and tomograms are available upon request from D. R. Tweeton, Twin Cities Research Center, Bureau of Mines, Minneapolis, MN.

FIELD TESTS IN LIMESTONE QUARRY

Experimental Method

Because an adequate data set was not obtained from the Bureau's on-site test facility, Bureau researchers gathered a more complete set of travel times at the Basic Materials, Inc., limestone quarry near Waterloo, IA. The test site, shown in figure 6, is mostly competent limestone, with some fractured zones. Inspection of the nearby pit wall allowed the Bureau to select borehole locations between major vertical fractures.

The source borehole, near the bush on the left side of figure 6, was 18.2 m from the receiver borehole, by the seated person near the right of the figure. The water injection borehole was between them, 6.1 m from the receiver borehole. All three boreholes were 18 m deep and 8.9 cm in diameter. The water levels before injection were 16.3, 12.5, and 12.7 m in the source, injection, and receiver boreholes, respectively. The source and receiver positions were from 2.4 to 12.2 m below the surface, all above the water table. There were 17 source positions and 17 receiver positions, directly across from each other. Position intervals were 0.61 m.

The source, shown in figure 7, was an air gun with a diameter of 6.4 cm and a 330-cm³ chamber run at 10.3 MPa (1,500 psi). This source provided adequate seismic energy, even in these dry boreholes. The receiver was the wall-locking geophone receiver system developed at the Bureau. The frequency transmitted through the limestone was centered at 200 Hz. The waves were recorded on a 24-channel seismograph. Each waveform was recorded with 959 samples (959 dots on the seismograph display) at a sampling frequency of 20,000 Hz.

Data were first collected when no water was being injected. After the preinjection tomographic data set was collected, water was injected into the middle borehole to raise its water level to 5.2 m below the surface. The water was maintained at that level by injecting about 2 L/h. Such a low flow rate indicates the low permeability of the rock. The water level in the other wells did not rise. The lack of a rise in the receiver borehole water level did not mean that the limestone near that borehole did not become saturated, only that the flow rate was not large enough to cause the water level to rise. After the water level in the middle borehole had stabilized overnight, a second set of tomographic data was collected using the same source and receiver positions as for the preinjection set.

The precision (measurement error) in the data⁷ was 0.05 ms, based on the sampling frequency. The accuracy was estimated to be 0.1 ms, based on repeatability.

Analysis and Results

The data were analyzed with a curved-ray tomographic program. The ray-tracing portion of the program, RAYPS, was written by Michael Weber at the University of Frankfurt

(currently at Seismologisches Zentralobservatorium, Grafenberg, Federal Republic of Germany).⁸ Chris Calnan of the University of Utah added algorithms for calculating the travel times for individual receivers. The Bureau added algorithms for tomographically calculating the distribution of seismic velocities that best fit the travel times.

The curved-ray tomographic program adjusts the seismic velocities at nodes and then calculates the derivatives of velocity in triangles with those nodes as corners. The derivatives of seismic velocities are used for calculating the bending that occurs in each triangle. A continuous velocity at the triangle boundaries was assumed. There were 16 nodes horizontally and 21 nodes vertically. The positions of source and receiver were centered between nodes at the edge of the grid. The node region was extended below the positions of the source and receiver because the paths tended to curve downward toward the high-velocity zones. In the bottom row of the tomograms, nodes for which there were no nearby paths were left uncolored. The only constraint applied in the analyses was to set the upper velocity limit to 4.2 km/s. Without that constraint, the velocity at some nodes became unreasonably high.

Five iterations were required to obtain the reconstructions shown in figures 8 and 9. There was little qualitative change from iteration 4 to iteration 5, although average velocities rose slightly as paths were bent more to higher velocity zones. Additional work will be required to improve the stability of the curved-ray tomographic program. Ideally, the program should be able to run for an unlimited number of iterations without departing from a solution. However, when more than 10 iterations were allowed, the program began to give unreasonably high velocities when it was not constrained.

The reconstruction using preinjection data (fig. 8) showed a high-velocity zone sloping upward from near the bottom on the source side to the receiver side, with a low-velocity zone below the high-velocity zone on the receiver side at a depth of 10.4 to 11.6 m. The indicated presence of a low-velocity zone, implying fractures, is consistent with the tendency of the geophone receiver to become stuck at 10 to 11 m depth. Unfortunately, there were no cores or driller's logs for these boreholes.

The reconstruction using data taken during injection is shown in figure 9. To facilitate comparing velocities before and during injection, figure 10 displays the velocities during injection minus the velocities before injection. The mound in the water level is where it should be. The former low-velocity zone near the bottom on the receiver side acquired a higher velocity. This change is consistent with the flow of water from the injection well into the fractured zone, increasing the seismic velocity. An artifact appeared near the top of the grid, where the calculated velocity increased, although no water was injected there. Also, there were some nodes where the calculated velocity decreased. That decrease is probably an artifact of the

⁷Data are available upon request from D. R. Tweeton, Twin Cities Research Center, Bureau of Mines, Minneapolis, MN.

⁸The ray tracing was based on Weber's 1986 Ph.D. thesis, "Die Gauss-Beam Methode zur Berechnung Theoretischer Seismogramme in Absorbierenden Inhomogenen Medien: Test und Anwendung" ("The Gaussian Beam Method for Calculating Theoretical Seismograms in Absorbing Inhomogeneous Media: Investigation and Application").



Figure 6.—Limestone quarry field test site.



Figure 7.—Air gun seismic source.

reconstruction. Thus, modification of the program to allow the application of appropriate constraints will be required. The use of sonic borehole logs to determine the seismic velocities at the sides of the ray-path region could help to reduce artifacts.

The average velocity was 2.92 km/s before injection and 3.24 km/s during injection, a 10-pct difference. That change is consistent with previous laboratory experiments. It was less than the decrease observed in the Bureau site, probably because the quarry rock was less fractured. The travel times decreased by an average of 5 pct. The reason that average velocities increased by a greater percentage than travel times

decreased is that paths became longer as they bent. Paths bent more during injection because of higher velocity contrasts.

The results were encouraging. The mound was located in the right position, and most of the pattern of changing velocities was consistent with what was known about the site. The data from this experiment and from others demonstrate that the seismic velocity does increase when dry rock becomes wet under field conditions. Extreme drying or saturating, as in the laboratory, was not necessary to observe the effect, and the increase in velocity was large enough to measure with commercial seismic equipment.

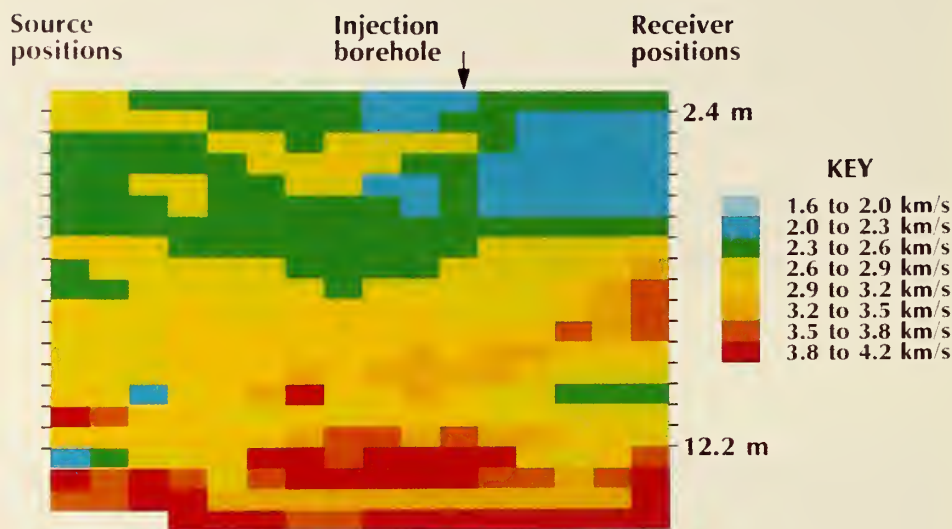


Figure 8.—Seismic velocities before water injection.

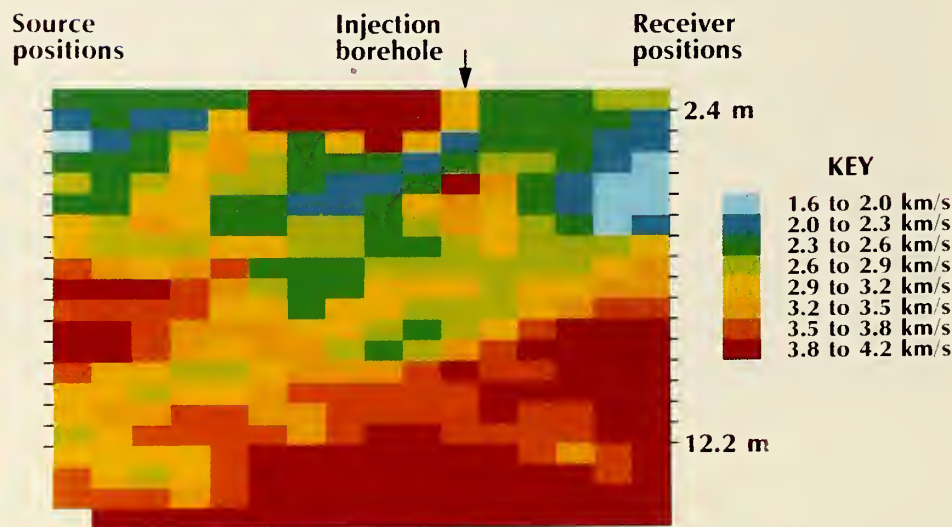


Figure 9.—Seismic velocities during water injection.

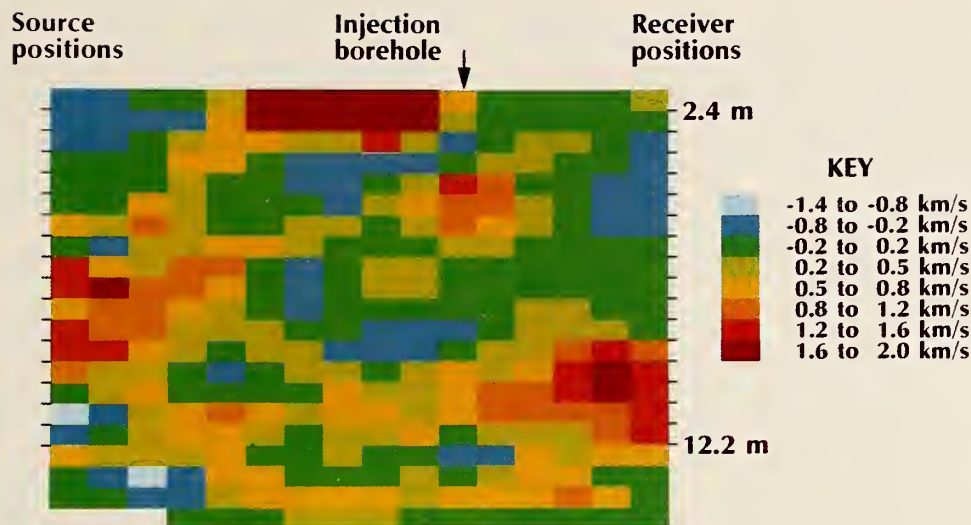


Figure 10.—Seismic velocities during injection minus velocities before injection.

ELECTRICAL AND ELECTROMAGNETIC METHODS

The seismic method is not expected to be able to distinguish between leach solution and ground water. Therefore, leach solution injected below the water table will not be located by seismic methods. A method that responds to a different physical property, such as electrical conductivity, is necessary. The conductivity of acidic solutions will usually be higher than the conductivity of the surrounding ground water, thus providing a good target for electrical and electromagnetic methods. Electrical and electromagnetic methods under consideration include magnetotellurics (MT), galvanic resistivity, ground-penetrating radar (GPR), and frequency- and time-domain electromagnetics (FEM and TEM). All of these methods have been used in locating and monitoring contaminated ground water, chemical spills, and brine plumes and are believed to be applicable to leach solution detection as well. These methods are discussed in many references, such as Telford (19) and the recent comprehensive publication by the Society of Exploration Geophysicists (20).⁹

One of the most formidable problems associated with the detection of leach solution is the great depth at which in situ mining can occur. Some porphyry copper deposits in the desert southwest are over 600 m deep, and many geophysical systems cannot penetrate that far. In addition, complications arise from the presence of conductive overburden, such as saturated clays, which greatly attenuate electrical signals. Thick conductive overburden blankets various parts of the desert southwest where in situ copper mining is being considered. Therefore, geophysical systems having deep-penetrating capabilities are desirable for this application.

CSAMT is a modification of earlier MT and audio-magnetotelluric (AMT) methods. MT and AMT are natural source systems relying on electrical energy provided by thunderstorms and related natural activity. Although these natural source systems have great depth-penetrating capabilities, they suffer from high noise levels caused by un-

predictable electric fields. Consequently, natural source methods would not have the resolution necessary to detect a small, deeply buried leach zone.

CSAMT, on the other hand, uses an artificial or controlled current source, a transmitter with a long grounded dipole or ungrounded loop. Current may be injected at any of several frequencies, depending on the depth desired. This eliminates the dependence on unpredictable electric fields. Although CSAMT does not have the penetration capabilities of its natural source counterparts, it does give more accurate information.

Like any electrical or electromagnetic method, CSAMT suffers from noise introduced by geologic inhomogeneities, conductive overburden, and multiple or nonuniform targets. A leach zone, whose conductivity, shape, size, and distribution may vary considerably within the ore deposit, may be difficult to recognize in one area and easy to recognize in another. Use of several loop or dipole sizes and frequencies, as well as use of other methods, may be necessary to delineate subsurface solutions. Further studies will be necessary to determine the feasibility of CSAMT for this application, despite its earlier failure in the detection of carbonate leach solution (1).

Galvanic resistivity, or simple resistivity, is a direct-current system in which a voltage is measured as the result of an applied current. This voltage, which is a function of the subsurface resistivity, can be used to map changes in lithology, structure, and soil or ground water conductivity. Resistivity electrode arrays can be configured in several different ways, depending on the survey objective, and can be used in both surface and borehole modes.

Conventional resistivity methods, which depend on a point source (single electrode) of current, have limited depth of penetration. A modified method, known as focused resistivity, holds more promise, since its depth of penetration is much greater.

Focused resistivity relies on three equidistant collinear point sources, rather than a single source. Current flowing from the outside sources tend to constrain, or focus, the center

⁹A second volume, to be published by the Society of Exploration Geologists, will give a method-by-method treatment of principal electromagnetic techniques.

current source. This "focused current" is directed vertically downward and propagates a great distance before diffusing, giving it deep-penetrating capabilities. This system has been used successfully by Southwest Research Institute for detecting subsurface cavities or voids¹⁰ and could be applied to deep conductive bodies, such as leachate plumes, as well. Focused resistivity techniques may also be used in a surface or borehole configuration.

Theoretically, the borehole configuration would be the most advantageous, since operation takes place closer to the target, but in practice, the borehole configuration is less desirable. Cased holes and conductive leachate fluids in the holes severely reduce the usefulness of borehole techniques. Prior to surveying under such conditions, the casing may have to be pulled out or, more likely, the cased portion left unsurveyed and fluids pumped out of the hole. Some research agencies are investigating three-dimensional resistivity measurements using sources that may penetrate through casings.

GPR operates by transmitting a high-frequency pulse into the surrounding medium. If this pulse impinges upon an electrical conductor, such as a leach solution plume, part of the pulse will be reflected back to the receiver. In a surface configuration, GPR can be used to map the lateral extent of the plume. In a borehole configuration, a radial picture of media surrounding the borehole is obtained.

Unfortunately, GPR, like all high-frequency systems, suffers from wave attenuation when used in conductive ground. Wave attenuation reduces signal amplitude to such an extent that deep exploration with such systems becomes impossible. One way to reduce the problem is to decrease the frequency, but this tends to accentuate dispersion and distortion effects. In addition, power consumption is higher and resolution becomes poorer. GPR may be useful in shallow or short-range detection and monitoring applications.

Electromagnetic methods use an alternating current in a coil to produce a primary magnetic field. Secondary fields are

generated when eddy currents swirl inside a subsurface conductor. These secondary fields produce voltages in receiving coils, which are then recorded by the operator. Measurements can be made in either time or frequency domains. TEM measurements are made by shutting off the primary field, then recording the voltage as a function of time. FEM measurements are made while the primary field is on. The secondary field is then measured as a function of frequency. The receiver may be on the surface or in a borehole.

TEM is better suited for deep detection of leach solution plumes than is FEM, especially if conductive overburden blankets the area. Conductive overburden tends to diminish the signals going to and coming from the target. To measure the small signals generated by a conductor under a thick conductive overburden, it is desirable to increase the signal-to-noise ratio. This may be accomplished in several ways. First, TEM measures the secondary field in the absence of the primary field, thus eliminating the need for exact transmitter-receiver-coil separation and orientation. Exact coil separation is necessary in FEM surveys to compensate for the primary field strength, allowing the much weaker secondary field to be measured. Second, TEM measurements facilitate stacking, a process by which a measurement is repeated many times in a short period of time, which helps to reduce random background noise. Finally, TEM systems employ large transmitter coils, sometimes several hundred meters across, resulting in enormous magnetic dipole moments and hence a large signal. Large dipole moments become extremely important when investigating conductive bodies buried under thick conductive cover.

Computer simulations are being conducted to investigate whether this method has sufficient sensitivity. Preliminary calculations are encouraging, and Bureau researchers believe that TEM is the most amenable system for locating conductive leach solution plumes. After computer simulations are completed, the most promising methods will be selected for experimental evaluation.

SUMMARY

Bureau researchers believe that certain geophysical techniques are promising as an aid to detecting probable flow paths and monitoring leach solution during in situ mining. The most promising techniques are seismic cross-hole tomography and electromagnetic induction. Seismic cross-hole tomography has the potential for being a practical procedure because of the relatively low cost of the equipment and the large penetration

distance. The Bureau has developed components of a seismic cross-hole tomography system. Analyses of experimental data indicate that the method is promising, but more research will be required to minimize mathematical artifacts unrelated to the presence of leach solutions. Electromagnetic induction is promising in principle because of its ability to detect a good conductor in the presence of poor conductors. GPR may be useful for detecting fracture zones and leach solutions near the borehole, but its limited penetration range will prevent it from being used for monitoring.

¹⁰Work done under Bureau contract H0245005.

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ENHANCED WELL-DRILLING PERFORMANCE WITH CHEMICAL DRILLING-FLUID ADDITIVES

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ABSTRACT

The U.S. Bureau of Mines has investigated the use of chemical additives to enhance drilling performance, for which conventional hard-rock drilling and in situ borehole development are target applications. Laboratory diamond core drilling tests were performed on Sioux Quartzite, Westerly Granite, Minnesota taconite, and Tennessee marble, using solutions of cationic additives, nonionic polymer, and acid-base buffered solutions as drilling fluids. Penetration improvements ranged from 88 to over 650 pct and bit life improvements from 56 to over 400 pct with chemical additive solution concentrations that produced a zero surface charge (ZSC) on the rock. The ZSC condition was found to be the most important factor in improving drilling performance. The nonionic polymer was found to be the best additive in the laboratory drilling tests and the most appropriate for field use because the ZSC condition could be obtained over a wide range of solution concentrations (10 to 125 ppm). Field validation tests of rotary tricone drilling with polymer on Minnesota taconite increased penetration rate by about 69 pct and bit life by about 26 pct. Application of this enhanced drilling phenomenon to well drilling for in situ mining should be pursued because of the potential for time and cost savings.

INTRODUCTION

As part of an overall program to increase minerals extraction efficiency, the Bureau of Mines has investigated the use of chemical additives in drilling fluids to enhance drilling performance in hard rock. Watson (1)⁵ surveyed and summarized 60 yr of literature-reported results of using chemical additives in rock fragmentation processes. In that 60-yr period, some researchers reported no improvements when drilling with additives in the drilling fluid while others reported significant benefits, especially when drilling with additive solutions that create an electrochemical point of zero charge (PZC) or zero surface charge (ZSC) on the surface of the rock being drilled. Some noteworthy successes include Reh binder's work (2), in which drilling efficiency increases of up to 60 pct were reported; Shepherd's work (3), in which slightly lower increases in drilling efficiencies were obtained; and Westwood's

work (4), which indicated that drilling efficiency could be improved well over 100 pct. The survey showed that, while the use of chemical additives in the drilling fluid was sometimes beneficial, little scientific reasoning was provided in the literature to explain either the presence or absence of beneficial effects when these chemical additives were used.

Intrigued by these reports, the Bureau designed and conducted laboratory drilling studies to sort out the disparities of earlier research and to establish and understand the necessary boundary conditions for drilling performance improvement under ZSC conditions. This paper describes the laboratory research conducted to determine the boundary conditions of the enhanced ZSC-controlled drilling phenomenon and to determine the applicability of the phenomenon with respect to (1) rock type, (2) bit type, and (3) surface charge (zeta potential) modifier. Also included are results from several initial field validation tests of ZSC-controlled drilling, which suggest that drilling with the recommended additive could produce significant drilling performance improvements. One potential application for the use of additive-assisted drilling is in the development of in situ mining deposits, for which considerable time and cost savings could be realized.

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⁵Italic numbers in parentheses refer to items in the list of references at the end of this paper.

ROCK MATERIALS USED IN TESTS

All rock samples for laboratory drilling tests were wire sawed into 15-cm cubes. Rock fragments of the same samples were ground to minus 149 μm for zeta potential measurements and chemical analyses. Previous Bureau reports have given detailed descriptions of Sioux Quartzite and Westerly Granite, including their physical properties (5), and Minnesota taconite and Tennessee marble (6). Briefly, Sioux Quartzite is a homogeneous, fine-grained, metamorphosed sandstone that has a relatively fracture-free structure, is comprised mainly of quartz grains, and averages 98.5 pct silica. Samples were obtained from a quarry in southwestern Minnesota. Westerly Granite is a light-gray, fine-grained, equigranular granodiorite, containing about 65 pct feldspars, 25 pct quartz, 9 pct micas, and about 1 to 2 pct accessory minerals. The Westerly Granite used in this research was obtained as a single 1,500-lb slab from an active quarry in Bradford, RI. Min-

nesota taconite is a dark green to gray, fine-grained, metasedimentary rock consisting of chert, magnetite, hematite, siderite, and the following silicates: minnesotaite, greenalite, stilpnomelane, and amphibole. The taconite has mildly undulating black bands of magnetite that occur in irregular layers, which account for the overall 27 to 30 pct iron content. The taconite was provided by Erie Mining Co., located near Hoyt Lakes, MN. Tennessee marble (quarry trade name) used in this investigation came from an active mine in the Holston Limestone formation in the Great Valley of east Tennessee (7). The formation is an essentially unmetamorphosed, coarsely crystalline limestone of Middle Ordovician age. The formation shows a range of colors from light gray to pinks and red to dark brown. Chemically, the rock is about 97.4 pct calcium carbonate, with trace amounts of other alkali minerals.

WATERS USED IN TESTS

Four waters were used in the drilling tests: distilled, deionized water (DDIW), tap water, mine pond water, and mine well water. Chemical analyses of these waters are given in a previous report (6).

DDIW was prepared by distilling Minneapolis tap water in a high-capacity 200-L still and then passing it through standard and ultrapure ion-exchange cartridges. The pH of the DDIW was in the range 5.3 to 6.0, and the conductivity measured from 0.3 to 0.5 $\mu\text{mho/cm}$.

The Minneapolis tap water used in these tests had an average pH value of 7.3 to 7.7, and the conductivity was about 200 $\mu\text{mho/cm}$.

Both the mine pond water and mine well water were obtained from Erie Mining Co. in northern Minnesota. The mine pond water, used as a source of water at Erie Mining during the spring, summer, and fall, is obtained from a pond at the bottom of one of the company's open pits. The mine well water, used as a source of water during the winter, is pumped from a well on the property. The pH of the pond water ranged from 7.0 to 7.5, while the pH of the well water was 7.9 to 8.0. Both had conductivities from 500 to 700 $\mu\text{mho/cm}$.

SURFACE CHARGE CONSIDERATIONS

The surface charge of most rocks in water around neutral pH is negative. If sufficient amounts of surface charge modifiers are added to the drilling fluid, such as inorganic salts, cationic surfactants, cationic polymers, and a special nonionic polymer, the surface charge can be neutralized,

resulting in the ZSC condition. The zeta potential gives a measure of rock surface charge. Relationships between zeta potential, double-layer theory, and surface charge have been discussed in a previous report (5).

ZETA POTENTIAL MEASUREMENTS AND ZSC CONCENTRATIONS

The procedure used for measuring zeta potentials and determining the ZSC concentration is described in detail elsewhere (5). Using a commercial zeta reader, zeta potentials are determined for the rock particles in water (DDIW, tap, or mine water) while concentrations of additive solutions are increased.

The zeta reader operates on the principle of electrophoretic mobility, whereby the speed of a particle in an electric field is proportional to its surface charge. The apparatus employs a video display to monitor particle movement in the electric field. When the speed of a moving grid line on the video display is matched to that of the particle, the zeta potential of that particle is shown on a digital readout.

For the drilling experiments tested, zeta potentials were determined for each rock type in water alone and in water with a series of concentrations of each additive. This is done to generate the range of negative and positive zeta potential values needed to graphically determine the concentration at which the zeta potential or surface charge of the rock particles is zero. It is important to determine each water-rock combination because ZSC concentrations were found to be substantially different for each rock tested and depend both on the chemical additive and type of water used.

DRILLING SYSTEM

The drilling apparatus employed in this investigation is shown in figure 1. A detailed description of the mechanical and electronic components of the drilling apparatus and the drilling procedure are given in a previous report (5). Briefly, drilling tests were performed on 15-cm rock cubes using a 1.12-kW drill press, fitted with a water swivel, and either a 16-mm-OD (10-mm ID) diamond-impregnated coring bit or a 16-mm tungsten carbide water-cooled spade bit.

The diamond-impregnated coring bits were rotated at a speed of 100 rpm under a thrust of 150 kg; the spade bits were also rotated at a speed of 100 rpm but under a lower thrust of 60 kg. Drilling fluid was flushed through either drill at a rate of 150 mL/min. Drilling was done perpendicular to the bedding plane of rocks where applicable.

The coring bit matrix was 100 pct cobalt (powder) sintered to a Rockwell C hardness of 18 to 21. The diamonds in the matrix were quoted by the manufacturer to be minus 425 μm plus 300 μm , although the size range of the diamonds was actually minus 1,000 plus 250 μm . The tungsten carbide spade bits were used only to drill the Tennessee marble. These bits featured replaceable, resharpenable blades with two waterways. Assembled, they resemble a machinist's straight-fluted-type drill bit of 16 mm diameter. The tip had a 125° included angle, a 125° chisel angle, and a 5° to 8° negative axial rake angle.

The diamond coring drill bits were sharpened by briefly drilling into a superduty fireclay brick (53 pct silica, 42 pct alumina) to produce a sharpness level corresponding to an initial average penetration rate for one hole of 4.5 mm/min in the test rock sample. The tungsten carbide blades were factory sharpened to a uniform degree; therefore, they were not resharpened in the laboratory. Drilling continued for a sequence of several holes until the average penetration rate for a hole had dropped to 2.0 mm/min in the test rock sample. The diamond coring drill bits were resharpened before another test began; the tungsten carbide blades were simply changed for the next test.

Data on total penetration and elapsed time of drilling in progressing from the sharp-bit state of 4.5 mm/min to the dull-bit state of 2.0 mm/min were recorded, added, and then used for test comparisons. The enhanced penetration performance of the additive, as compared with the appropriate baseline water, was calculated by

$$E_p = [C_p - W_p]/W_p \cdot 100, \quad (1)$$

where E_p = penetration effect for a test, pct,

C_p = total penetration for drilling with a given additive, mm,

and W_p = average total penetration for drilling with water alone, mm.

The bit life effect of the additive, as compared with the baseline water, was calculated by

$$E_t = [(C_t - W_t)/W_t] \cdot 100, \quad (2)$$

where E_t = bit life effect for a test, pct,

C_t = total time for drilling with a given additive, min,

and W_t = average total time for drilling with water alone, min.

The results of penetration and bit life effects for additive concentrations compared with their respective baseline water tests are described in the following section.

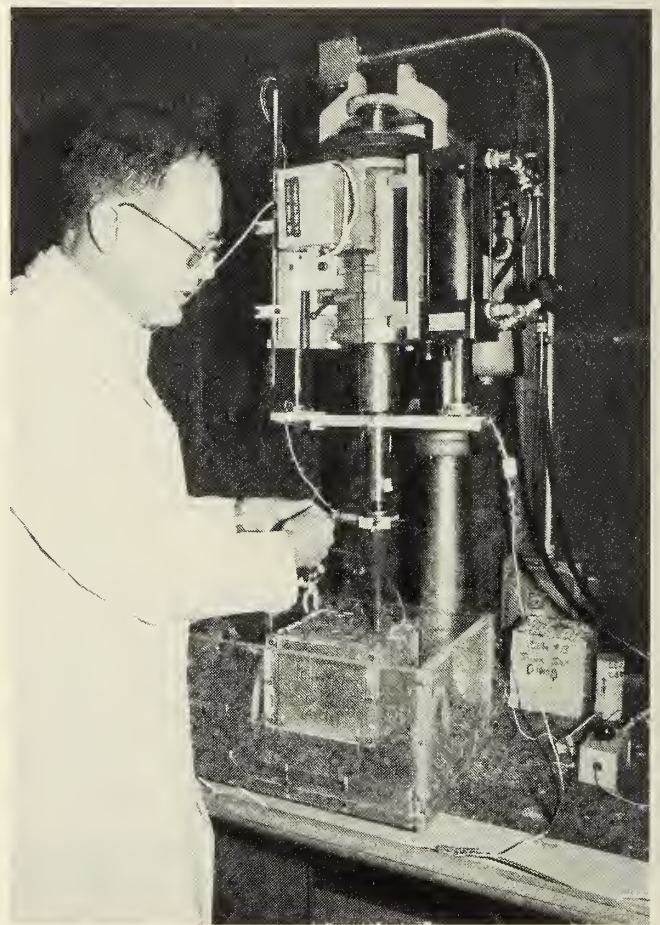


Figure 1.—Drilling apparatus.

LABORATORY DRILLING TEST RESULTS AND DISCUSSION

Results of laboratory drilling tests are presented in table 1 and figures 2 through 11.

Table 1.—Drilling results for all additives, percent improvement

Type of water, additive, and additive concentration	Penetration	Bit life	Type of water, additive, and additive concentration	Penetration	Bit life
SIOUX QUARTZITE			SIOUX QUARTZITE—Continued		
DDIW with AlCl_3 , mol/L:			DDIW with DTAB, mol/L:		
1.0×10^{-8}	-33.53	-33.76	3.0×10^{-4}	-32.92	-27.39
1.0×10^{-7}	40.12	44.69	9.8×10^{-4}	118.18	99.56
5.0×10^{-7}	98.50	98.71	1.5×10^{-3}	111.36	108.41
6.0×10^{-7}	103.83	85.12	DDIW with TTAB, mol/L:		
7.0×10^{-7}	62.87	35.02	5.0×10^{-6}	-34.79	-33.11
8.0×10^{-7}	61.99	40.96	9.0×10^{-6}	-5.82	-7.83
1.0×10^{-6}	28.60	23.13	1.7×10^{-5}	-.19	-4.75
3.0×10^{-6}	17.21	22.28	1.85×10^{-5}	1.08	-4.31
5.0×10^{-6}	52.47	50.31	2.0×10^{-5}	13.31	-3.14
7.0×10^{-6}	35.81	34.17	2.2×10^{-5}	-17.47	-11.64
1.0×10^{-5}	79.64	73.23	2.8×10^{-5}	-45.20	-45.60
5.0×10^{-5}	60.32	44.36	5.0×10^{-5}	-35.11	-33.51
1.0×10^{-4}	50.40	54.55	7.23×10^{-5}	87.84	87.02
DDIW with CaCl_2 , mol/L:			9.0×10^{-5}	-6.86	22.60
1.0×10^{-5}	.87	13.81	DDIW with HTAB, mol/L:		
1.0×10^{-4}	53.23	36.01	5.0×10^{-7}	9.27	1.26
1.0×10^{-3}	28.32	24.83	9.0×10^{-7}	71.67	54.57
5.0×10^{-3}	87.32	73.23	1.6×10^{-6}	87.70	55.94
7.0×10^{-3}	32.60	5.30	3.0×10^{-6}	-10.22	-10.80
9.0×10^{-3}	68.12	52.85	5.0×10^{-6}	-3.94	-3.27
1.1×10^{-2}	96.53	72.67	Tap water with PAA, ppm:		
2.0×10^{-2}	-14.18	5.74	0.10	-16.54	-15.52
3.0×10^{-2}	-7.46	-1.49	0.15	47.99	29.18
4.0×10^{-2}	1.42	5.30	0.20	209.17	183.14
5.0×10^{-2}	-17.18	-6.59	0.25	334.03	187.22
7.0×10^{-2}	-3.70	10.39	0.50	174.83	124.49
1.0×10^{-1}	1.56	20.58	1.0	26.82	22.02
DDIW with NaCl, mol/L:			Tap water with PEO, ppm:		
9.0×10^{-2}	16.52	5.30	1.0	66.16	45.55
1.0×10^{-1}	14.86	5.30	3.0	429.65	270.58
2.0×10^{-1}	115.03	76.35	7.5	348.56	235.69
3.0×10^{-1}	-3.50	-6.59	12.5	363.65	235.90
5.0×10^{-1}	2.22	15.49	125.0	387.60	330.05
DDIW with ZrCl_4 , mol/L:			Tap water at pH of 3.8		
1.0×10^{-7}	33.32	40.12	with AlCl_3 , mol/L:		
5.0×10^{-7}	-9.57	-1.49	7.0×10^{-7}	45.37	16.07
9.0×10^{-7}	39.54	35.87	1.0×10^{-4}	46.20	49.78
1.4×10^{-6}	96.29	67.71	1.15×10^{-4}	46.27	20.23
5.0×10^{-6}	4.48	-8.29	1.18×10^{-4}	80.51	48.97
1.0×10^{-5}	43.37	30.77	1.6×10^{-4}	80.74	66.31
1.0×10^{-4}	-46.33	-36.31	1.7×10^{-4}	95.82	34.06
DDIW with $\text{Al}(\text{NO}_3)_3$, mol/L:			1.8×10^{-4}	113.63	59.59
1.0×10^{-7}	-14.34	-5.01	1.9×10^{-4}	97.02	76.94
3.0×10^{-7}	84.46	55.16	2.0×10^{-4}	33.06	6.47
4.3×10^{-7}	98.09	63.50	3.0×10^{-4}	-26.82	-29.30
5.0×10^{-7}	-24.32	-16.35			
1.0×10^{-6}	11.38	18.82			

See footnotes at end of table.

Table 1.—Drilling results for all additives, percent improvement—Continued

Type of water, additive, and additive concentration	Penetration	Bit life	Type of water, additive, and additive concentration	Penetration	Bit life
WESTERLY GRANITE			MINNESOTA TACONITE—Continued		
DDIW with AlCl_3 , mol/L:			Mine well water with		
1.0×10^{-7}	18.66	0.83	Percol 402, ppm:		
3.0×10^{-7}	62.05	67.28	0.40	105.66	92.30
5.0×10^{-7}	133.75	92.72	¹ 0.64	174.98	141.40
6.0×10^{-7}	104.50	73.92	1.0	– 7.73	– 3.48
¹ 7.0×10^{-7}	154.66	109.00	6.4	9.90	22.34
1.0×10^{-6}	– 17.45	– 8.46	Buffered pH solutions, pH units:		
1.5×10^{-6}	– 16.81	– 22.70	7.0	31.00	54.70
¹ 2.0×10^{-6}	164.92	136.23	6.0	138.95	129.15
3.0×10^{-6}	– 36.42	– 29.23	¹ 5.5	278.63	218.14
5.0×10^{-6}	– 19.79	– 22.07	3.5	19.43	49.29
7.0×10^{-6}	66.72	– 52.01	Mine pond water with PEO, ppm:		
1.0×10^{-5}	– 16.31	– 16.92	3.0	107.80	130.66
MINNESOTA TACONITE			7.5	246.76	220.14
DDIW with AlCl_3 , mol/L:			¹ 12.5	647.52	421.28
7.0×10^{-7}	56.24	42.68	125.0	661.94	603.67
9.0×10^{-7}	53.93	28.25	TENNESSEE MARBLE		
1.0×10^{-6}	70.20	71.15	DDIW with AlCl_3 , mol/L:		
¹ 1.2×10^{-6}	143.45	104.46	4.0×10^{-7}	15.94	39.65
1.4×10^{-6}	135.55	104.07	1.6×10^{-6}	25.07	50.18
1.7×10^{-6}	22.48	6.14	¹ 1.9×10^{-6}	50.09	98.18
2.0×10^{-6}	34.17	37.50	2.3×10^{-6}	84.25	146.09

¹ Point of zero charge concentration.² Compared with baseline water of same pH 3.8 level.

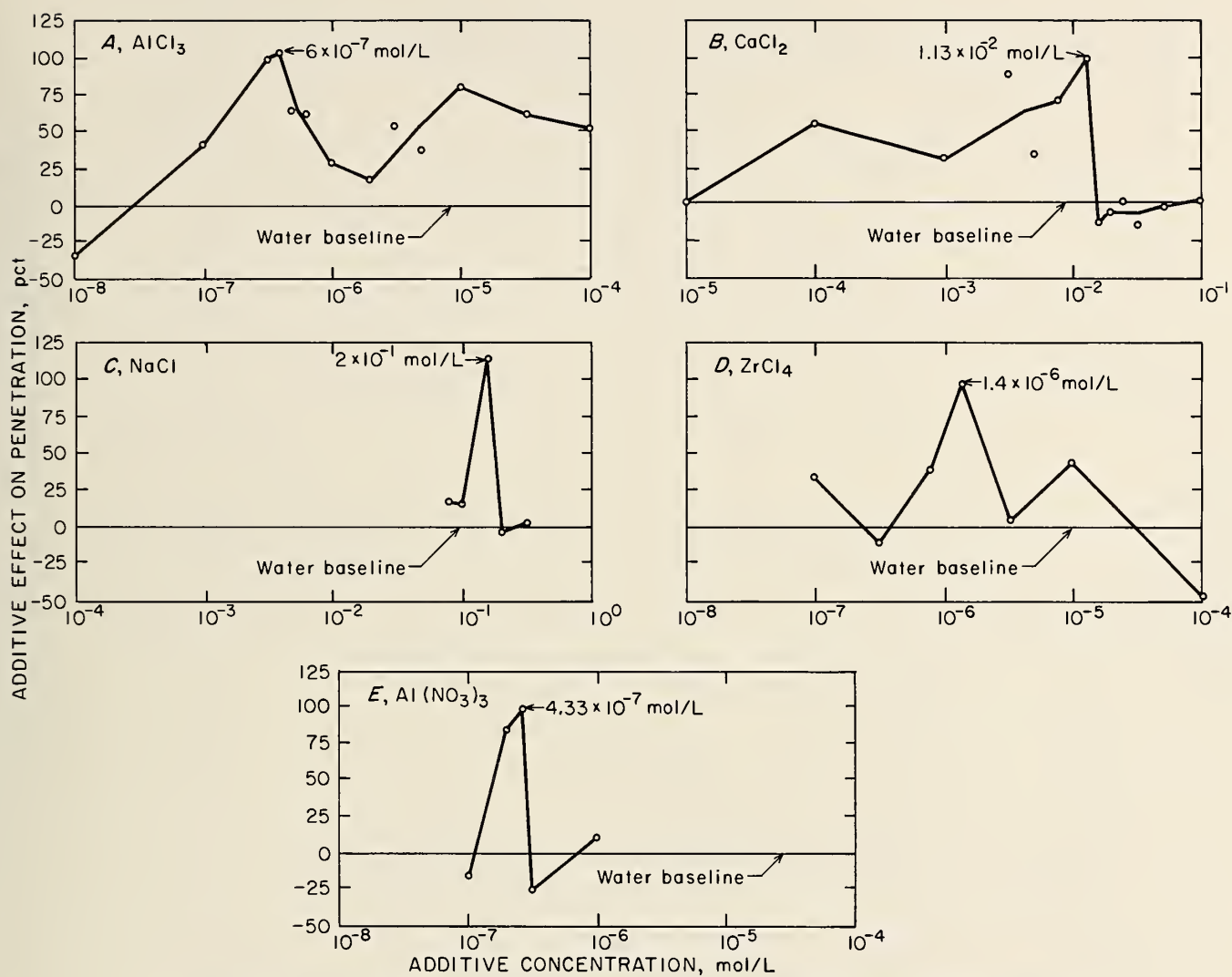


Figure 2.—ZSC control of drilling penetration in Sioux Quartzite with inorganic salts in DDIW.

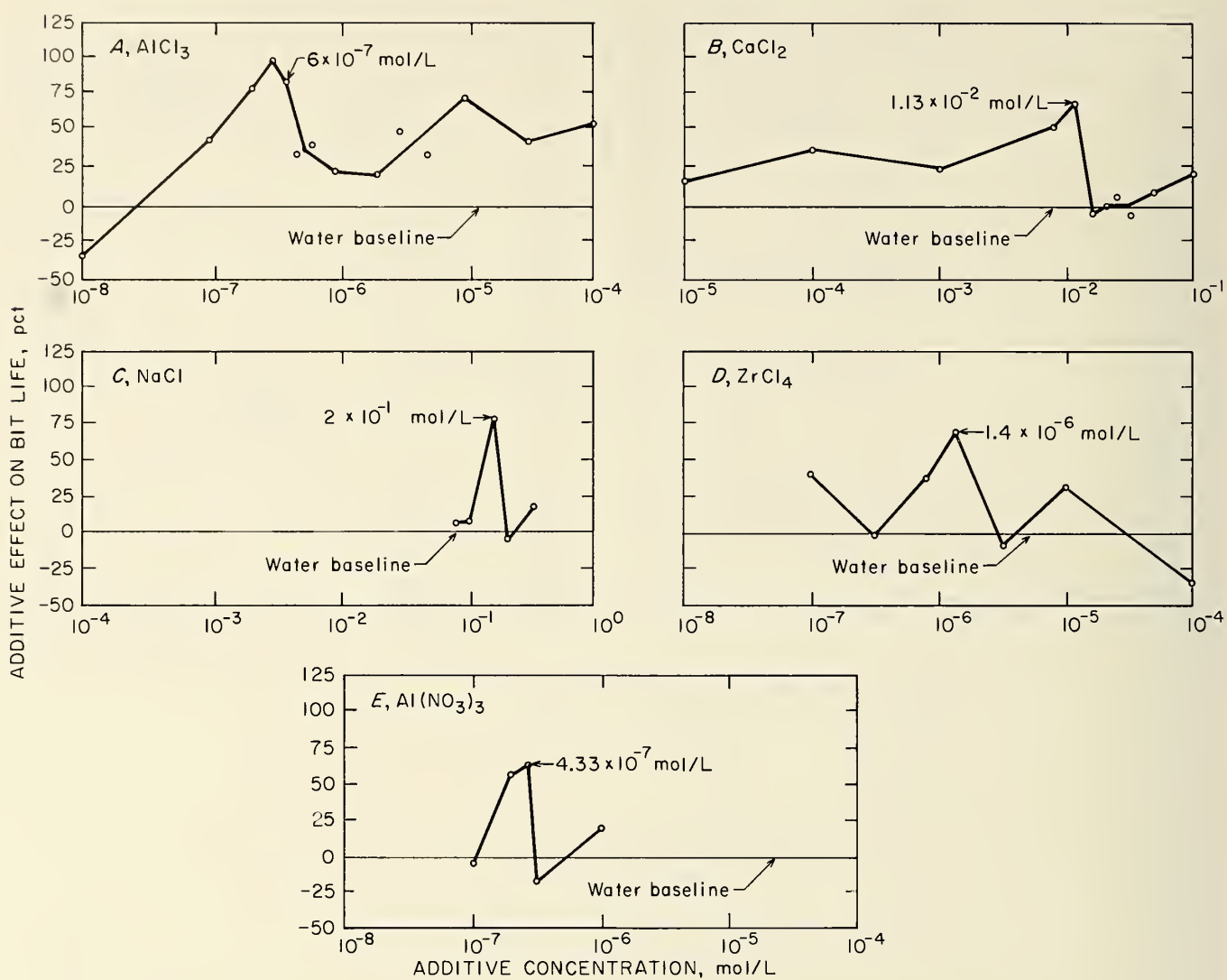


Figure 3.—ZSC control of bit life in drilling Sioux Quartzite with inorganic salts in DDIW.

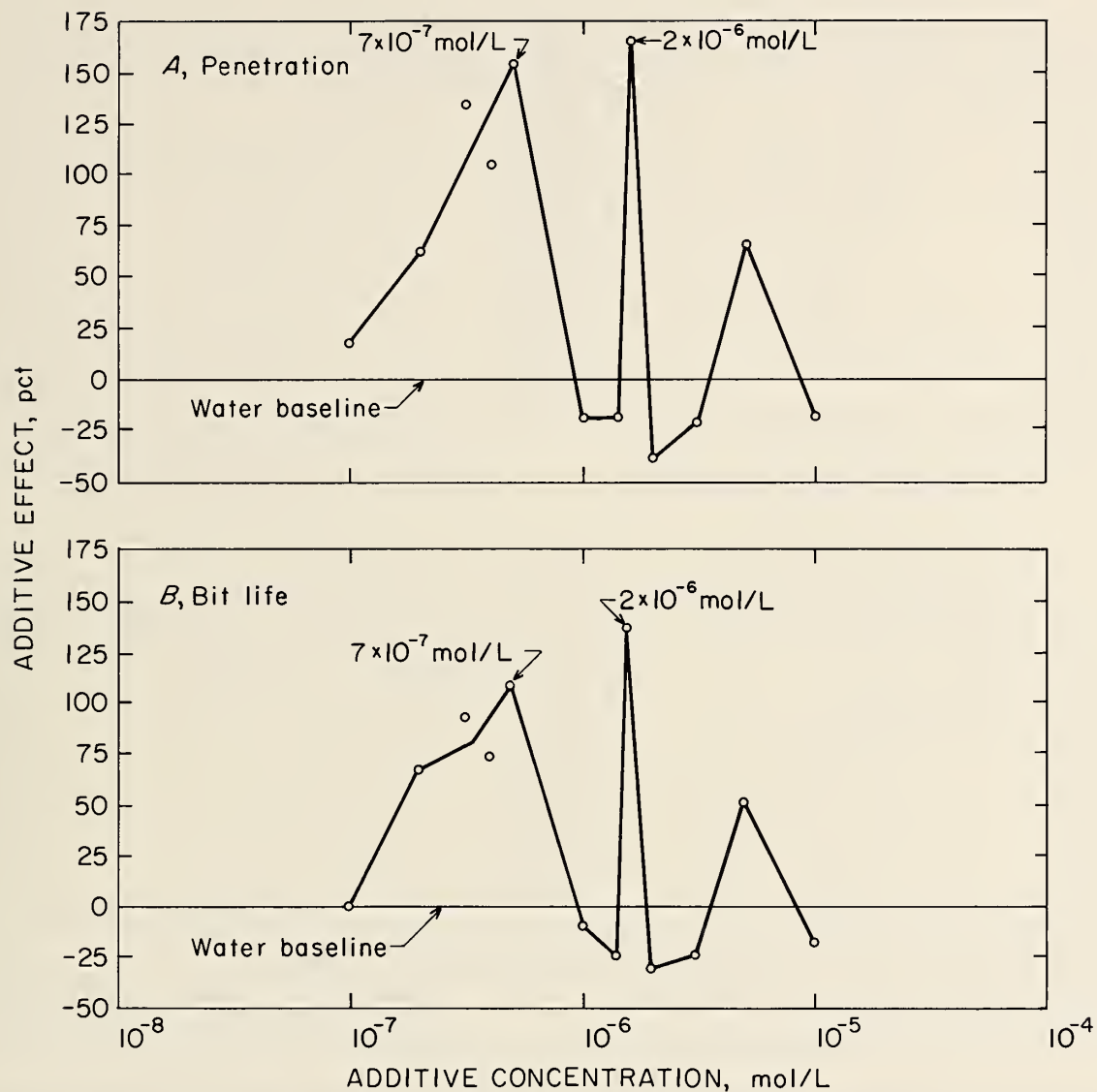


Figure 4.—ZSC control of drilling penetration and bit life in Westerly Granite with aluminum chloride in DDIW.

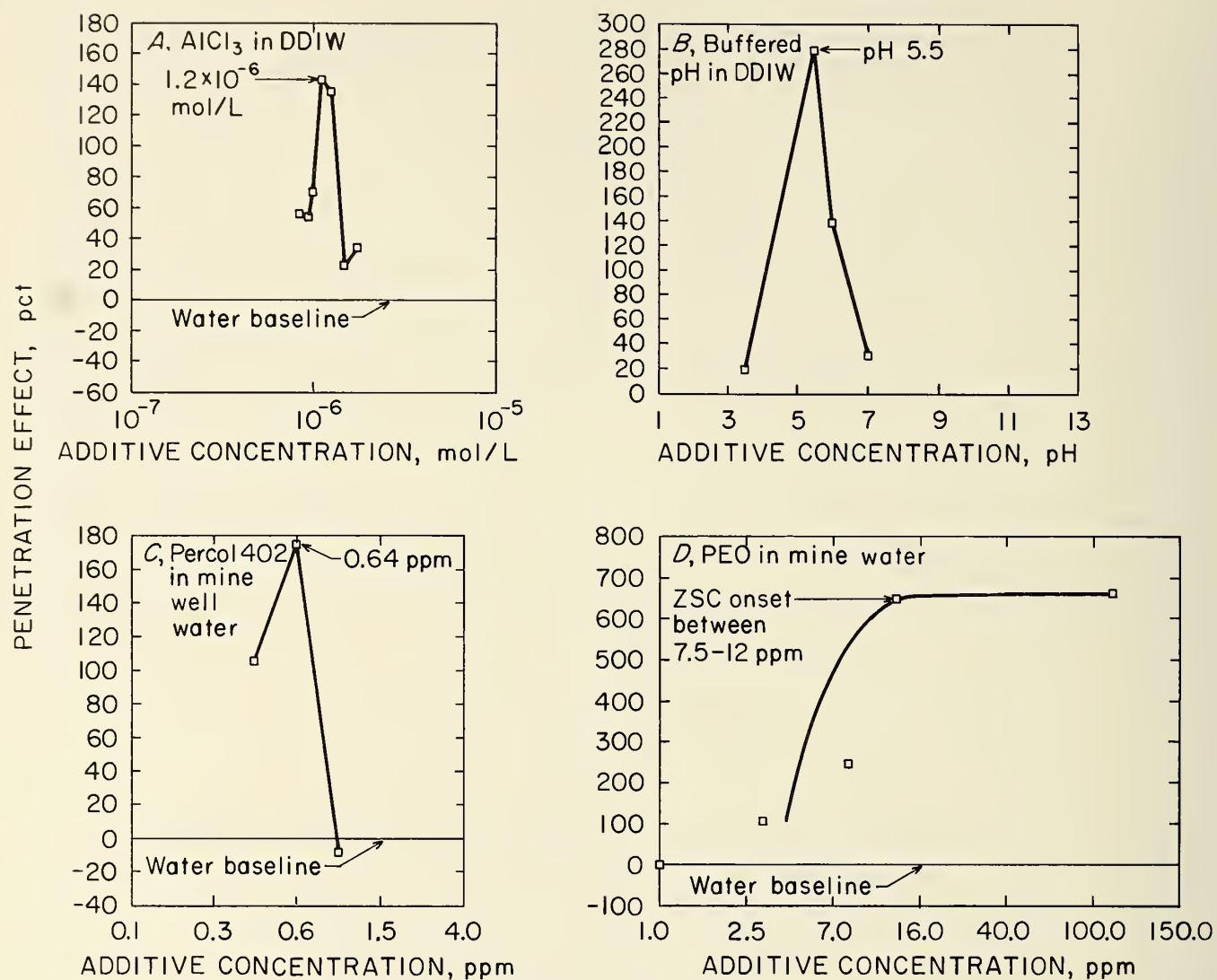


Figure 5.—ZSC control of drilling penetration in Minnesota taconite with aluminum chloride, acid, Percol 402, and PEO in various waters.

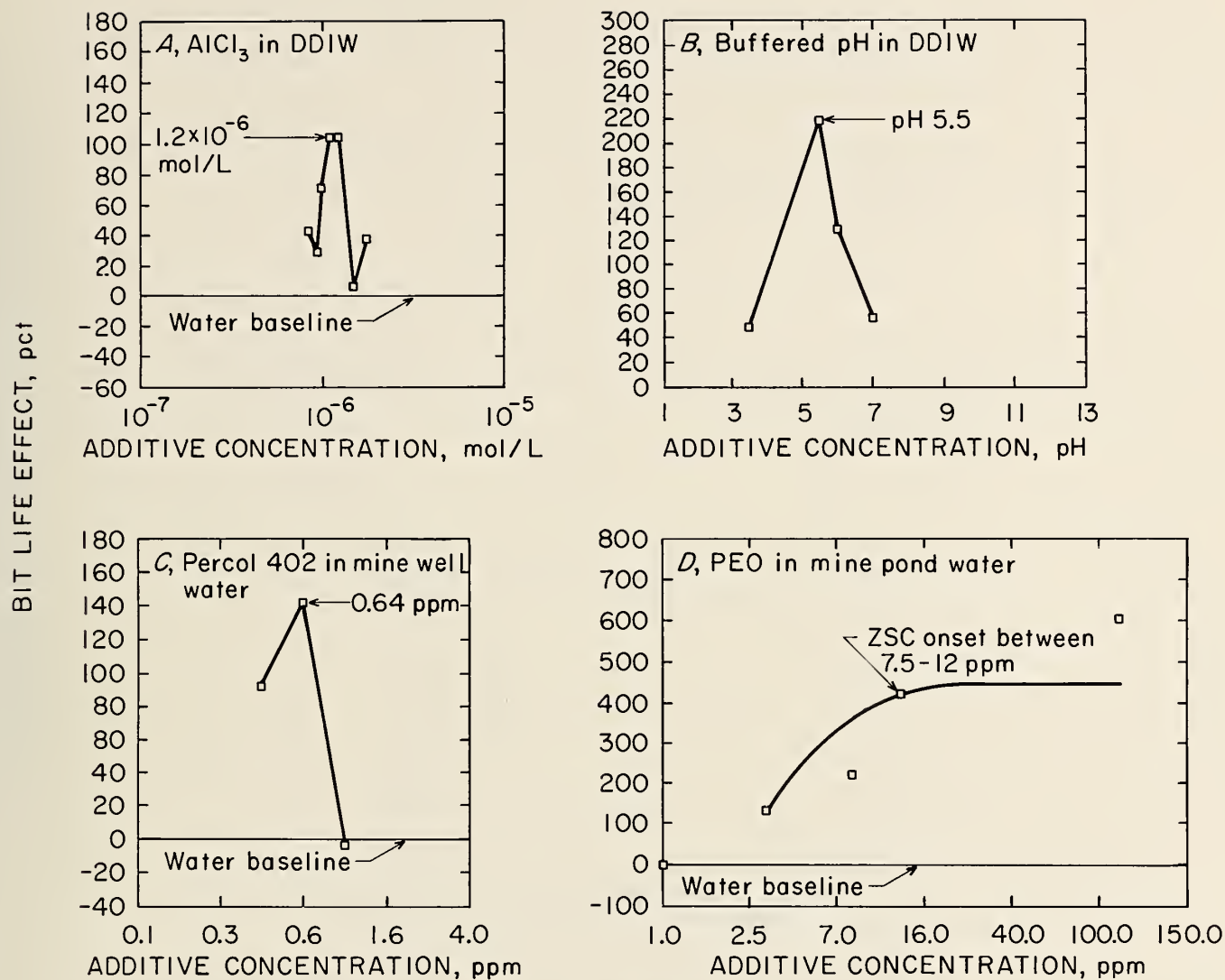


Figure 6.—ZSC control of bit life in drilling Minnesota taconite with aluminum chloride, acid, Percol 402, and PEO in various waters.

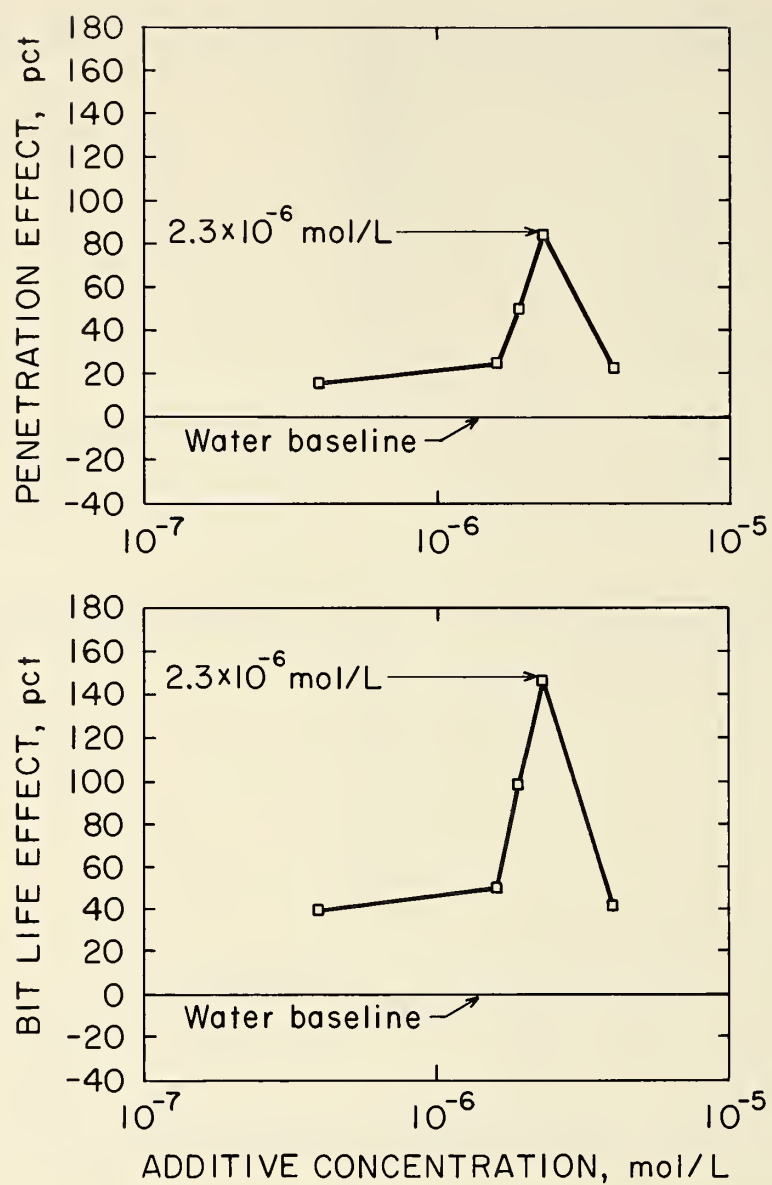


Figure 7.—ZSC control of drilling penetration and bit life in Tennessee marble with aluminum chloride in DDIW.

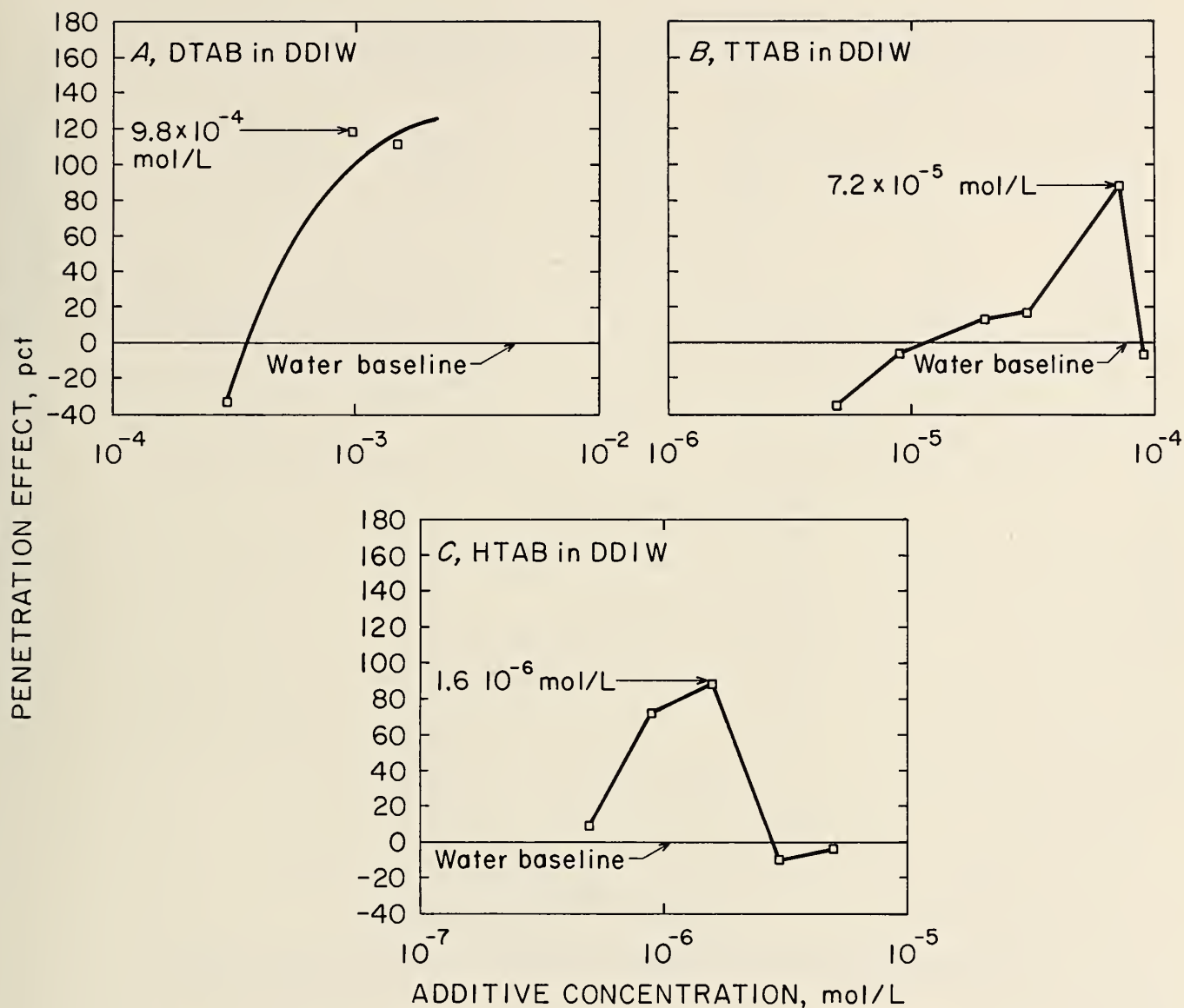


Figure 8.—ZSC control of drilling penetration in Sioux Quartzite with cationic surfactants in DDIW.

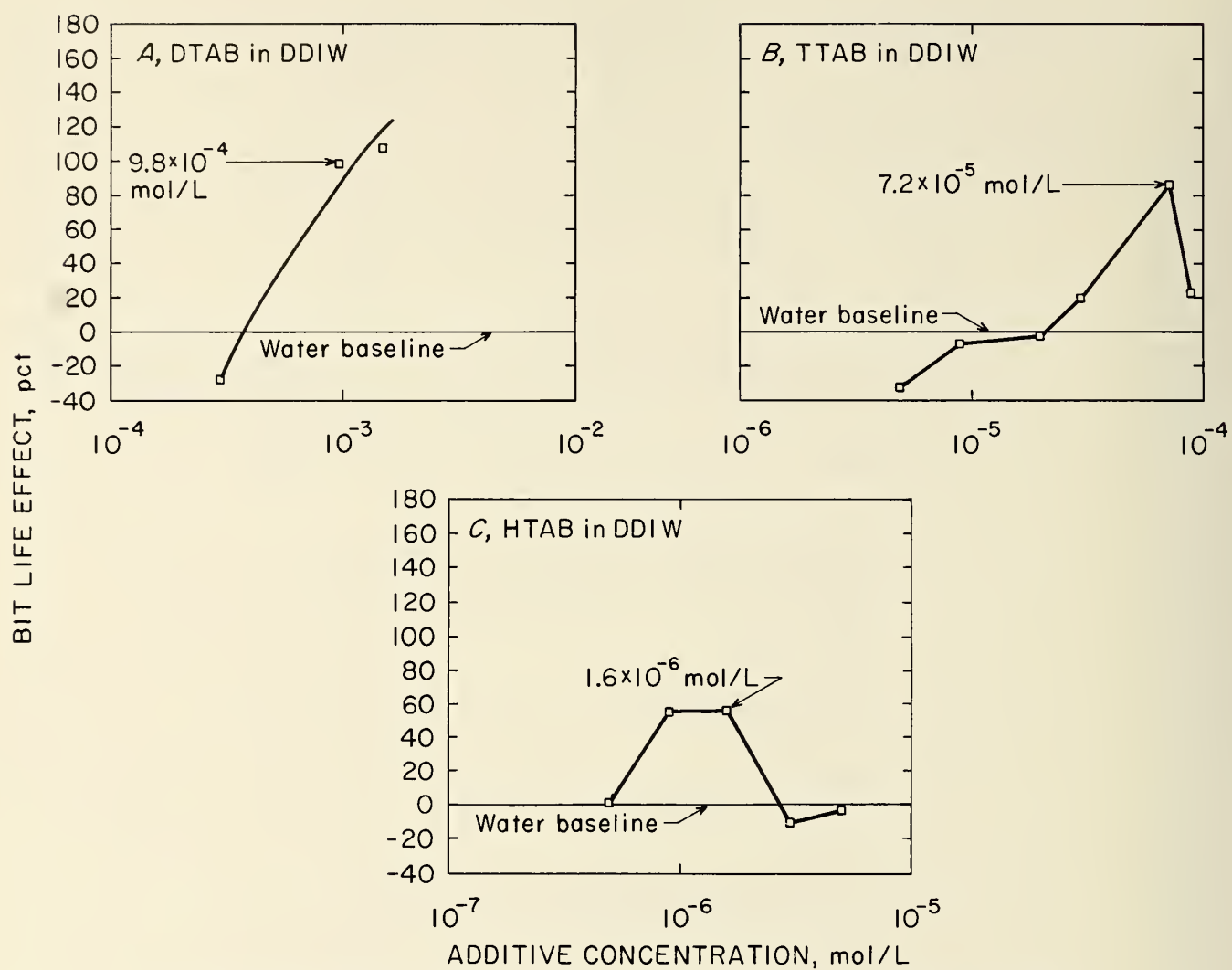


Figure 9.—ZSC control of bit life in drilling Sioux Quartzite with cationic surfactants in DDIW.

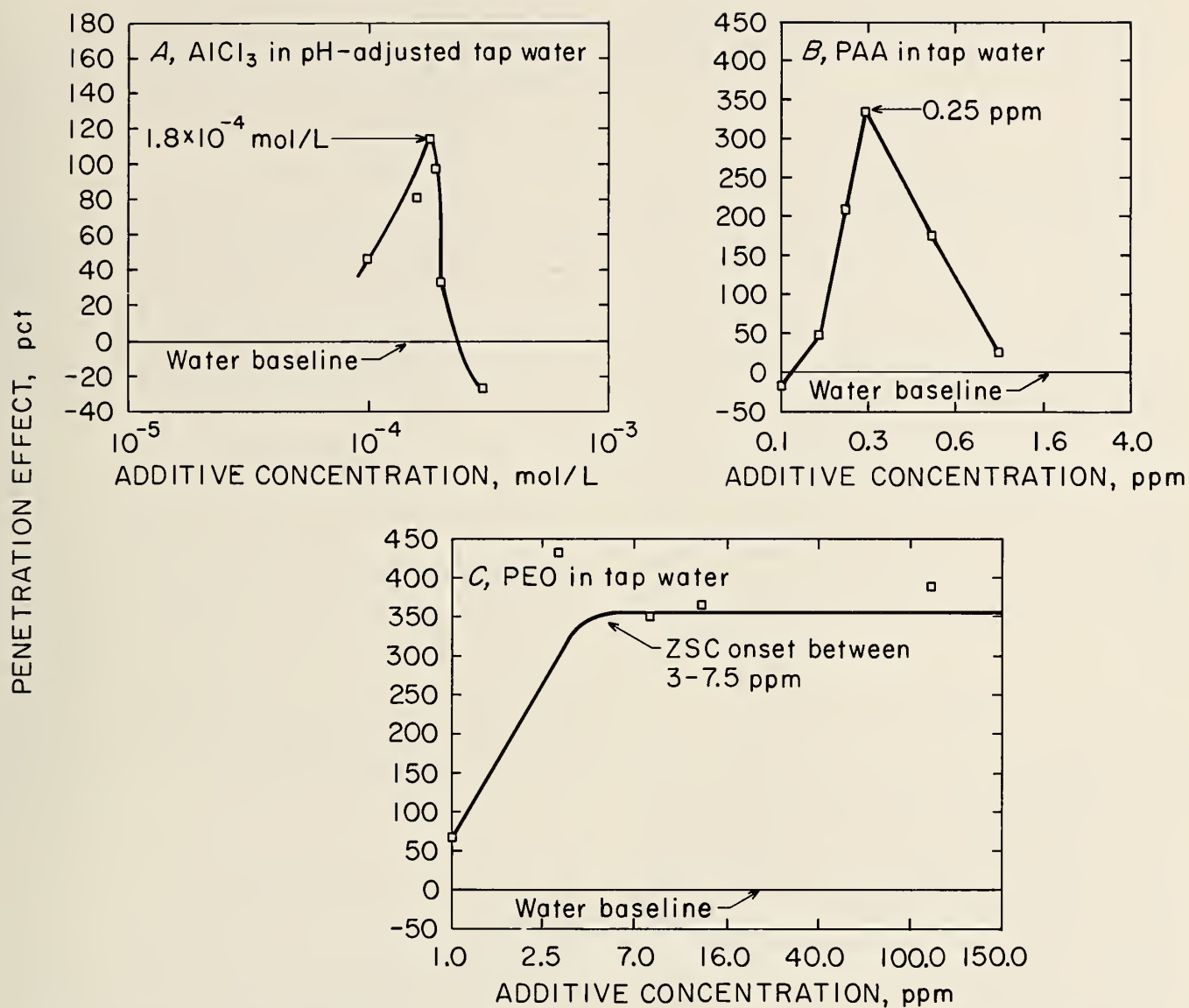


Figure 10.—ZSC control of drilling penetration in Sioux Quartzite with aluminum chloride, PAA, and PEO in various waters.

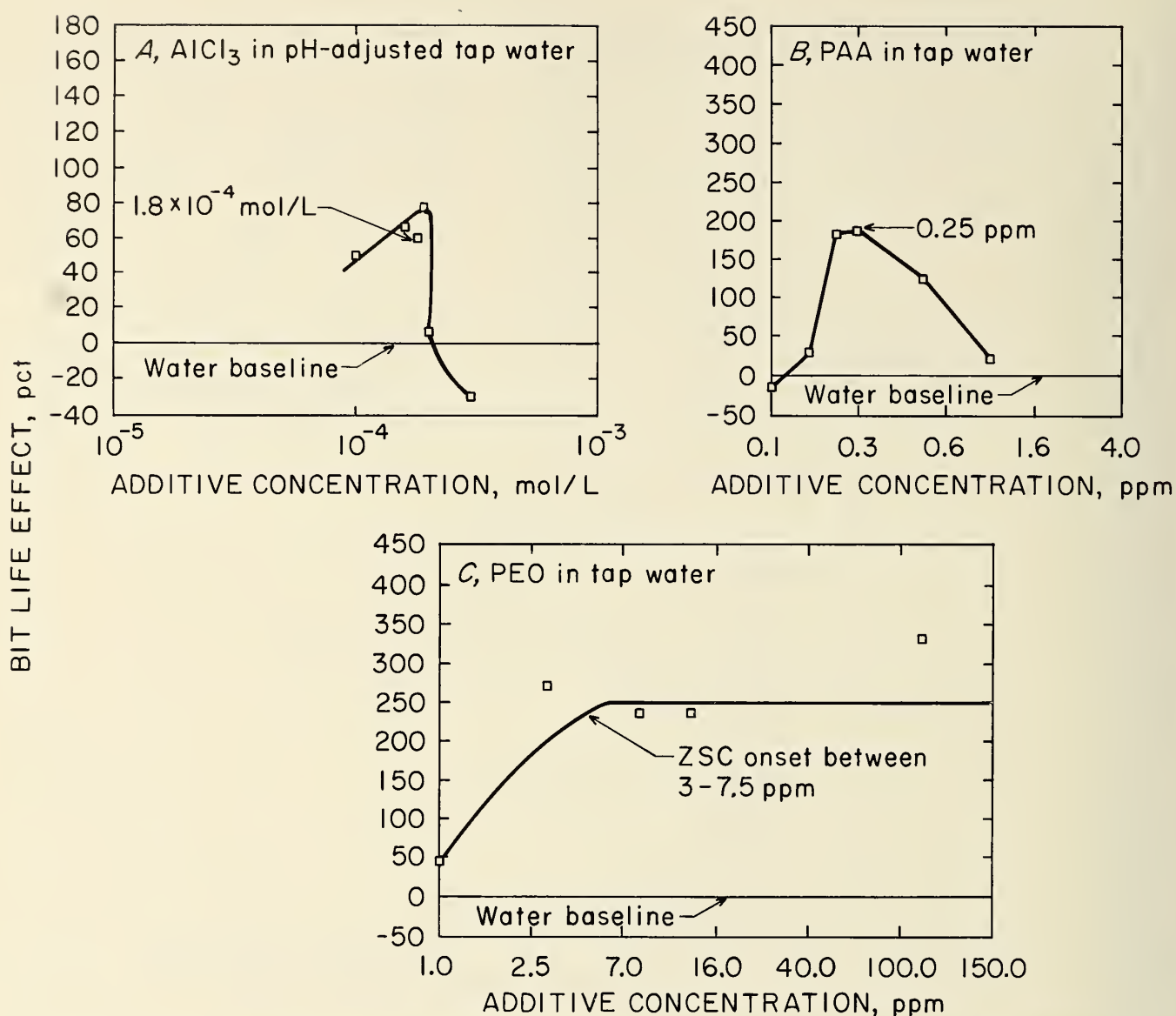


Figure 11.—ZSC control of bit life in drilling Sioux Quartzite with aluminum chloride, PAA, and PEO in various waters.

EFFECT OF ROCK TYPE ON ZSC-CONTROLLED DRILLING

Determination of the rock's response to ZSC-controlled drilling involved testing of rocks of high-, medium-, and non-silicate content. Sioux Quartzite and Westerly Granite were chosen as representative high-silicate rock types, Minnesota taconite was chosen as the representative medium-silicate rock type, and Tennessee marble (Holston Limestone) was chosen as the representative nonsilicate rock type.

Sioux Quartzite

Penetration and bit life effects for drilling Sioux Quartzite with various concentrations of aluminum chloride (AlCl_3) solutions, compared with drilling using DDIW, are given in

table 1. Drilling with aluminum chloride at the ZSC concentration of 6×10^{-7} mol/L produced a maximum increase in penetration of 104 pct and a near maximum in bit life of 85 pct (figs. 2A-3A).

Westerly Granite

Figure 4 shows plots of penetration and bit life effects for drilling Westerly Granite as a function of aluminum chloride concentration, while table 1 lists the penetration and bit life effects for each aluminum chloride solution concentration tested. Maximums in penetration effect of 155 and 165 pct, respectively, and in bit life effect of 109 and 136 pct were obtained when drilling with two concentrations of aluminum chloride. These maximums correspond to the ZSC concentration for the two main components of the granite: quartz at

7×10^{-7} mol/L and feldspars at 2×10^{-6} mol/L aluminum chloride. The origin of the third maximum is not understood at this time. It was concluded from these results and those for Sioux Quartzite that ZSC-controlled drilling is applicable to rocks containing high amounts of silicate minerals.

Minnesota Taconite

Penetration and bit life effects comparing drilling of Minnesota taconite with various concentrations of aluminum chloride to drilling with DDIW are given in table 1. Drilling with aluminum chloride at the ZSC concentration of 1.2×10^{-6} mol/L produced simultaneous maximum increases in penetration and bit life of 143 and 104 pct (figs. 5A-6A). It was concluded from these results that ZSC-controlled drilling is also applicable to rocks containing moderate amounts of silicate.

Tennessee Marble

Penetration and bit life effects comparing drilling of Tennessee marble with various concentrations of aluminum chloride are given in table 1. The drag, or spade-type, bit was used only for drilling into this rock type. Drilling with aluminum chloride at 2.3×10^{-6} mol/L, very near the ZSC concentration of 1.93×10^{-6} mol/L, resulted in a peak increase in penetration improvement of 84 pct and a bit life extension of 146 pct (fig. 7).

It is concluded from the drilling results for Sioux Quartzite, Westerly Granite, Minnesota taconite, and Tennessee marble that ZSC-controlled drilling should be applicable to all rock types, whether they have high-, medium-, or no-silicate contents.

EFFECT OF BIT TYPE ON ZSC-CONTROLLED DRILLING

ZSC-controlled drilling enhancement with diamond-impregnated bits has been demonstrated for Sioux Quartzite, Westerly Granite, and Minnesota taconite. Drilling of Tennessee marble was done with a tungsten carbide spade-type bit to determine whether ZSC-controlled drilling was applicable to other bit types.

As noted by Westwood (4), the cutting mechanisms of the diamond coring and tungsten carbide spade bit types are sufficiently different. Conventionally, spade bits as described here are not used in rock drilling, but rather in the machine-tooling trades. However, if a rock is not too hard, it can be drilled with a spade bit, which works with a scraping or dragging motion. Often the depth of cut for sharp spade bits is greater per pass than diamond bit cutting, all other conditions being equal, but the bits wear down quickly.

The demonstration of successful ZSC-controlled drilling enhancement for both drag bit types, i.e., diamond drills and spade bits, strongly implies the applicability of ZSC-controlled drilling to these types of bits. It is furthermore suggested from these results that ZSC-controlled drilling should be applicable to all drag bit types and possibly to all drill bit types.

Drilling of Tennessee marble with the diamond-impregnated coring bit was also tried; however, no decline in drilling rate was observed, even after numerous test holes. This meant that any diamond drilling tests using sharp- and dull-bit states, as defined by this study, would take far too long to be practical in the test program.

EFFECT OF SURFACE CHARGE MODIFIER ON ZSC-CONTROLLED DRILLING

Inorganic Salts

Drilling tests on Sioux Quartzite were conducted using various concentrations of solutions of the chloride salts of aluminum (AlCl_3), calcium (CaCl_2), sodium (NaCl), and zirconium (ZrCl_4), as well as aluminum nitrate [$\text{Al}(\text{NO}_3)_3$], as drilling fluids. The resulting penetration and bit life effects are given in table 1 and plotted in figures 2 and 3. These figures show that maximum penetration and bit life improvements were obtained with ZSC concentrations of each of the five inorganic salts tested. Although higher ZSC concentrations were required for lower valence cations, the enhanced drilling performance was still obtained. On the basis of these drilling results and those for Westerly Granite, Minnesota taconite, and Tennessee marble with aluminum chloride, it is concluded that drilling with ZSC concentrations of inorganic salt solutions should result in improved penetration and bit life responses. Determination of the applicability of ZSC-controlled drilling to all types of surface charge modifiers required testing of previously tested rock types with other types of surface charge modifiers such as organic cationic surfactants, cationic polymers, and nonionic polymers with some cationic character.

Cationic Surfactants

Four cationic surfactants were tested as surface charge modifiers. Three of these surfactants were familiar quaternary ammonium salts: the 12-carbon dodecyltrimethyl ammonium bromide (DTAB), the 14-carbon tetradecyltrimethyl ammonium bromide (TTAB), and the 16-carbon hexadecyltrimethyl ammonium bromide (HTAB). The fourth surfactant was a commercially available low-molecular-weight cationic polymer, Percol 402. DTAB, TTAB, and HTAB drilling solutions were prepared in DDIW and tested on Sioux Quartzite; therefore, the drilling performances obtained with these additive solutions were compared with the baseline drilling performance for Sioux Quartzite with DDIW. Percol 402 drilling solutions were prepared in Erie Mining Co. mine well water and were tested on Minnesota taconite. Percol 402 drilling performance was therefore compared with the baseline drilling performance for Minnesota taconite with Erie Mining mine well water. Drilling tests were conducted with surfactant solution concentrations below, at, and above the ZSC concentration.

Penetration and bit life effects for drilling Sioux Quartzite with DTAB, TTAB, and HTAB in DDWI are given in table 1 and plotted versus surfactant concentration in figures 8 and 9. Drilling with DTAB at a concentration of 9.8×10^{-4} mol/L, which is very near the ZSC concentration of 9.4×10^{-4} mol/L, produced a maximum increase in penetration of 118 pct and a near maximum in bit life of 100 pct (figs. 8A-9A). Penetration effect improvements for TTAB and HTAB at their respective ZSC concentrations of 7.2×10^{-5} mol/L and 1.6×10^{-6} mol/L were both 88 pct, while bit life extensions were 87 and 56 pct, respectively (figs. 8B-9B, 8C-9C).

Penetration and bit life effects for Percol 402 are given in table 1 and plotted as a function of concentration in figures 5C and 6C, respectively. Because the exact equivalent molecular weight of Percol 402 was not known, solutions were made up in parts per million expressed as volume per volume. Drilling

with the very dilute solution of Percol 402, which produces the ZSC condition at 0.64 ppm, resulted in the best drilling performance: a 175-pct improvement in penetration effect and a 141-pct improvement in bit life.

It is concluded from the drilling results for cationic surfactants that drilling with ZSC-concentration solutions of these surfactants also gives rise to simultaneous maximum penetration effect and maximum bit life.

Acid-Base Solutions

Surface charge neutralization also occurs at the isoelectric point (IEP) that is achieved by adjusting the pH of the water with either acid or base, depending upon whether the IEP pH is lower or higher than the incipient pH of the water. For magnetite, the IEP pH is about 5.5. Therefore, drilling of Minnesota taconite was conducted with solutions whose pH's were above, at, and below the IEP pH value of 5.5. Penetration and bit life effects are given in table 1 and plotted as a function of pH in figures 5B and 6B. Buffered solutions were employed to ensure that the desired pH was maintained throughout the drilling test. This was important since the major chemical byproduct of rock drilling has been found to be hydroxide ions (OH^-). Monitoring the pH of both the influent and effluent streams when drilling with DDIW alone has shown that drilling raises the drilling fluid pH as much as 3 to 4 pH units. Drilling with a pH 5.5 buffered solution (acetic acid-sodium acetate) resulted in simultaneous increases of 279 pct in penetration and 218 pct in bit life compared with drilling using DDIW alone. One possible explanation for these much higher drilling performance improvements, which are about twice those for drilling Minnesota taconite with ZSC-concentration solutions of aluminum chloride, is that the ZSC condition is maintained throughout the test by the buffered solution. With the ZSC-concentration solution of aluminum chloride, this is probably not the case, as the hydroxide ions produced in drilling alter or vary the surface charge condition during the course of the drilling test. The results of these tests further point to the universal application of ZSC-controlled drilling irrespective of the type of additive employed to neutralize the surface charge.

Cationic Polymer

Because drilling particulates flocculate in a ZSC-concentration solution, flocculation was thought to be partially responsible for the enhanced drilling performance at the ZSC concentration. Therefore, two polymers (one cationic and one nonionic) that flocculate particulates primarily by a molecular bridging mechanism, and secondarily by a charge neutralization mechanism, were tested as drilling-fluid additives.

The cationic polymer used as a drilling-fluid additive was the water-soluble, high-molecular-weight polyacrylamide (PAA). Solutions of PAA were prepared in Minneapolis tap water for test drilling in Sioux Quartzite. The drilling performance of PAA was therefore compared with the baseline drilling performance of Sioux Quartzite in Minneapolis tap water. Solution concentrations of PAA below, at, and above the ZSC concentration were tested. Penetration and bit life effects for PAA are given in table 1 and are plotted as a function of concentration in figures 10B and 11B. As with other cationic additives, drilling with the ZSC concentration of PAA (0.25 ppm) produced simultaneous maximum increases in penetration and bit life effects, 334 pct and 187 pct, respectively. Also,

it should be noted that the PAA drilling results clearly indicate that flocculation is not responsible for enhanced ZSC-controlled drilling. While flocculation with PAA occurs at all concentration levels above 0.1 ppm, the enhanced drilling performance is achieved only at the ZSC concentration of 0.25 ppm. The results of these tests with the cationic polymer also indicate the universality of ZSC-controlled drilling irrespective of the type of additive employed to neutralize the surface charge.

Nonionic Polymer

The nonionic polymer used as a drilling-fluid additive was the water-soluble, high-molecular-weight polymer polyethylene oxide (PEO). Drilling solutions of PEO were prepared in Minneapolis tap water for testing Sioux Quartzite and in Erie Mining Co. mine pond water for testing Minnesota taconite. The drilling performance of PEO was compared with the respective baseline drilling performance for Sioux Quartzite in Minneapolis tap water and Minnesota taconite in Erie Mining mine pond water. Although PEO is available in a range of molecular weights from 100,000 to 6 million, drilling tests were conducted only with the 5-million-molecular-weight nonionic polymer because ZSC concentrations (parts per million) were found to be the same as for these other PEO molecular-weight varieties. Penetration and bit life effects for PEO in drilling Sioux Quartzite and Minnesota taconite are given in table 1 and are plotted as a function of concentration in figures 10C and 11C for Sioux Quartzite and 5D and 6D for Minnesota taconite.

PEO is an unusual surface charge modifier. Because it is nominally nonionic, it was expected not to neutralize the rock surface charge and, therefore, not to produce enhanced ZSC-controlled drilling. This expectation was due to the observation that zeta potential determinations of Sioux Quartzite particles in commercial nonionic surfactants, such as Tergitol NPX, Surfynol 465, and hexaethyl cellulose (a high-molecular-weight polymer), showed no effect on the surface charge of the Sioux Quartzite particles at any concentration regardless of molecular weight. Zeta potential determinations of Sioux Quartzite particles in solutions of ever-increasing concentration of PEO, however, showed that PEO, like the cationic additives (inorganic salts, cationic surfactants, and cationic polymers), alters the zeta potential from an initial negative value to zero. But, unlike the cationic additives, PEO does not produce a positive zeta potential at higher concentrations. Instead, at higher concentrations, the ZSC condition is maintained. PEO appears to have some cationic behavior in water, which allows it to neutralize the negative surface charge on rocks and produce the ZSC condition. In the absence of a charged rock surface, the cationic behavior, however, is not active and the rock surface charge remains neutral and is not made positive.

Comparison of performance graphs of the penetration and bit life effects as a function of PEO concentration for Sioux Quartzite or Minnesota taconite (figs. 10C-11C, 5D-6D) with similar plots using cationic surfactants (figs. 8-9, 5C-6C) and inorganic salts (figs. 2-3, 5A-6A) shows why PEO is the best drilling performance enhancer. For Sioux Quartzite, penetration effects of over 350 pct and bit life effects of over 235 pct were attained with PEO (from 3 to 125 ppm), and for Minnesota taconite, penetration effects of over 650 pct and bit life effects of over 420 pct were attained with PEO (from 12.5 to 125 ppm) compared with the much lesser penetration and bit life effects obtained with ZSC-concentration solutions of the cationic surfactants or inorganic salts. In addition, there is

a wide range of concentrations of PEO that produce a maximum penetration effect, compared with the single concentration of cationic additives that produces the maximum penetration effect.

It is concluded on the basis of the tests with PEO that this polymer is the best additive for accomplishing ZSC-controlled drilling. Not only does PEO produce the enhanced ZSC drilling phenomenon over a wide range of concentrations, it also produces much greater improvements in bit life and penetration. It is further concluded on the basis of the tests with PEO and the cationic additives that establishment of the ZSC condition is the most important factor in enhanced drilling.

EFFECT OF DRILLING-FLUID WATER ON ZSC-CONTROLLED DRILLING

DDIW was used as the baseline comparison water in initial drilling tests to clearly show any additive effect without additional spectator ions, such as those found in tap water or mine water, to mask the effect. Determining the applicability of ZSC-controlled drilling with regard to water type required testing of previously tested rock types with other water types, such as tap water and mine water. While the pH of the DDIW used in ZSC-controlled drilling research was constantly in the range 5.3 to 6.0, the pH of Minneapolis tap water ranged from 7.3 to 7.7 and the pH of Erie Mining Co. mine pond and well

waters ranged from 7.0 to 7.5 and 7.9 to 8.0, respectively. Therefore, in using aluminum chloride as the additive in tap water or in either of the mine waters, a pH adjustment of these waters was required before aluminum chloride was added, to prevent precipitation and flocculation of the Al^{+3} ions in solution as aluminum hydroxide $[Al(OH)_3]$. Acidifying the tap water to pH 3.8 with hydrochloric acid resulted in only a negligible decline in drilling performance on Sioux Quartzite, compared with plain tap water without the addition of acid.

Penetration and bit life effects for drilling Sioux Quartzite with aluminum chloride solutions made from acidified tap water compared with drilling with acidified tap water only are given in table 1 and graphically displayed in figures 10A and 11A. Drilling with the ZSC concentration of 1.8×10^{-4} mol/L aluminum chloride in pH-adjusted tap water resulted in a maximum increase of 114 pct in penetration and a near maximum of 60 pct in bit life compared with pH-adjusted tap water alone, thereby showing that enhanced drilling performance could also be obtained with aluminum chloride in acidified tap water.

It is concluded on the basis of these data, and results for drilling Minnesota taconite with Percol 402 and PEO solutions (figs. 5C-6C, 5D-6D) in mine well water and mine pond water, respectively, and Sioux Quartzite with PAA and PEO solutions in tap water (figs. 10B-11B, 10C-11C), that ZSC-controlled drilling is applicable to a wide range of drilling-fluid water characteristics.

FIELD DRILLING TEST RESULTS

MINNESOTA TACONITE

Field drilling tests are being conducted on taconite at various mining operations in the Mesabi Iron Range of northern Minnesota. One of these tests consisted of drilling numerous 50-ft blastholes, at 30- by 30-ft spacings, with 15-in-diameter rotary tricone bits on two adjacent benches of cherty taconite. Drilling performance using an air-water flushing mist was compared with performance using an air-PEO solution mist for drilling particulate removal and dust control. The first bench was drilled solely with water, in 1984, consuming six tricone bits, and has subsequently been blasted away. The second bench was drilled with water (first 40 holes) and PEO (next 100 holes) in 1987. Figure 12 shows the penetration rate results for this test. The average penetration rate for the 165 holes drilled with water in 1984 was 0.55 ft/min, while the average penetration rate for the 40 holes drilled with water in 1987 was 0.56 ft/min. The average penetration rate for the 100 holes drilled with PEO in 1987 was 0.93 ft/min. Assuming that the penetration rate for the section of bench 2 drilled with PEO in 1987 would be about 0.55 ft/min if the bench had been drilled with water, then drilling with PEO results in a 69-pct increase in penetration rate. Also, for water drilling, the average life for two bits was $2,745 \pm 12$ ft, while for PEO drilling the average life for two bits was $3,468 \pm 40$ ft: a 26-pct increase in bit life. An added benefit in using the air-PEO solution mist in rotary tricone drilling of taconite is that it drastically reduces the amount of dust generated.

It is concluded from these initial rotary tricone drilling tests that drilling with a mist of ZSC-concentration solutions of PEO in air results in increased penetration rates and increased bit life. Field drilling tests are continuing to further validate these initial results.

OTHER SILICATE ROCK SUITES

Field testing of PEO was also done under cooperative research programs with HDRK Mining Research Limited of Copper Cliff, Ontario, and Centre in Mining and Mineral Exploration Research (CIMMER) at Laurentian University in Sudbury, Ontario, under an umbrella agreement between the Canada Centre for Mineral and Energy Technology (CANMET) and the U.S. Bureau of Mines.

In the HDRK cooperative research program, 3 weeks of diamond core drilling and percussive drilling tests in quartzite

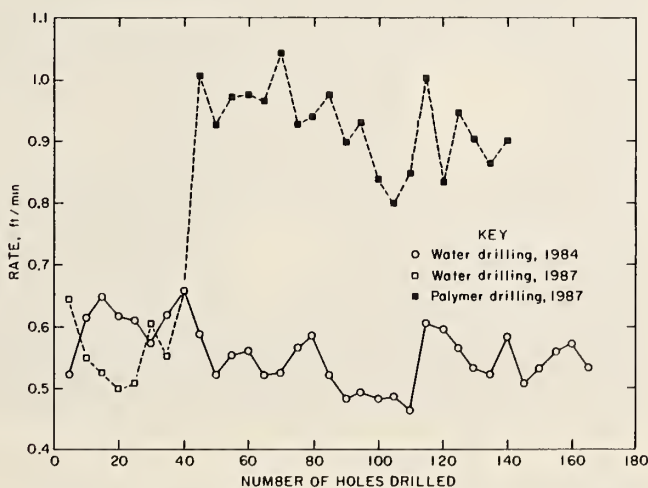


Figure 12.—ZSC control of penetration rate in field drilling of Minnesota taconite with PEO.

and quartz-mica schist were conducted at both underground and surface rock exposures under near-production conditions at the INCO Thompson Mine in Thompson, Manitoba. Diamond core drilling tests both on the surface and underground, under certain conditions, demonstrated improved bit life with ZSC-concentration PEO solutions. But, because of the large data scatter and small sampling population, the drilling results did not conclusively demonstrate the enhanced drilling performance with ZSC-concentration PEO solutions. The wide scatter in diamond core drilling bit life (footage drilled) can be attributed to inconsistencies in rock morphologies for holes drilled on the surface and variability in bit characteristics for holes drilled underground.

Drilling performance with ZSC-concentration PEO solutions was also determined in long-hole percussive drilling on the surface with 51-mm-diameter cross bits and underground with 51-mm-diameter button bits. Figure 13 shows an unused cross bit, a cross bit after drilling 100 ft in quartzite with 15 ppm PEO solution, and a cross bit after drilling 100 ft in the same rock with water alone. It can be seen that the wear on the PEO drilled bit is less than on the water drilled bit. To substantiate this visual observation, an optical comparator was used to sketch the 2-D wear profile trace for each of the bits. A typical comparator trace is shown in figure 14 for one of the carbide inserts from each of the three bits shown in figure 13. The area of these traces was measured and then used to calculate the difference in the wear of the carbide inserts. For the same distance of rock drilled, the cross bit used with polymer solution was worn 42.6 pct, whereas without polymer the wear was greater at 48.7 pct.

Typical comparator traces for one of the carbide button inserts from an unused button bit, a button bit used to drill 200 ft of quartz-biotite schist with 15 ppm PEO solution, and a button bit used to drill 200 ft of quartz-biotite schist with water alone are shown in figure 15. The area of these traces was measured and then used to calculate the difference in the wear of the carbide button inserts. For the same distance of

rock drilled, the button bit used with polymer solution was worn only 5.3 pct, whereas without polymer the wear was 12.1 pct, more than twice as much, showing a significant improvement using polymer solution.

In terms of penetration rate, tests using both the cross bit and button bit percussive drills showed no appreciable difference in average penetration rates when the use of polymer solution was compared with use of water alone as the drilling fluid. Cross bits advanced at about 1.9 to 2.2 ft/min, button bits at 1.8 ft/min in the respective rock faces.

Therefore, it is concluded from these data that drilling with ZSC concentrations of PEO solutions extends the life of both cross and button percussive bits, in spite of the lack of an improvement in penetration advance. Should these percussive bits be drilled to exhaustion, about three to five times longer than tested, it is expected that the polymer solution would further retard the carbide wear, permitting longer usage time per bit and longer intervals between carbide regrinds.

In the CIMMER cooperative research program, Longyear Canada, under the guidance of staff from Laurentian University and the Bureau, tested PEO as a drilling-fluid additive in exploration core drilling in granite from 3,000 to 5,000 ft below the surface, using BQ-size core bits. There was no measurable change in instantaneous penetration rate; however, bit life (total footage drilled) was improved 2.3- to 3.1-fold when drilling with PEO solutions, compared with drilling using water alone. Also, drill string vibration was reduced when PEO was used, compared with Dromus B soluble oil for this lubrication purpose. In addition, below 4,000 ft, the diamond-impregnated bits remained continually sharp; they did not have to be periodically sharpened by lowering the drill rotational speed momentarily (increasing torque) to abrade away the dull surface diamonds and expose a new layer of sharp diamonds. It is concluded from these data that bit life is extended and drill string vibration reduced in diamond core drilling using ZSC concentrations of PEO solution.



Figure 13.—Three cross bits. From left to right: unused bit, bit after drilling 100 ft quartzite with PEO, and bit after drilling 100 ft quartzite with water.

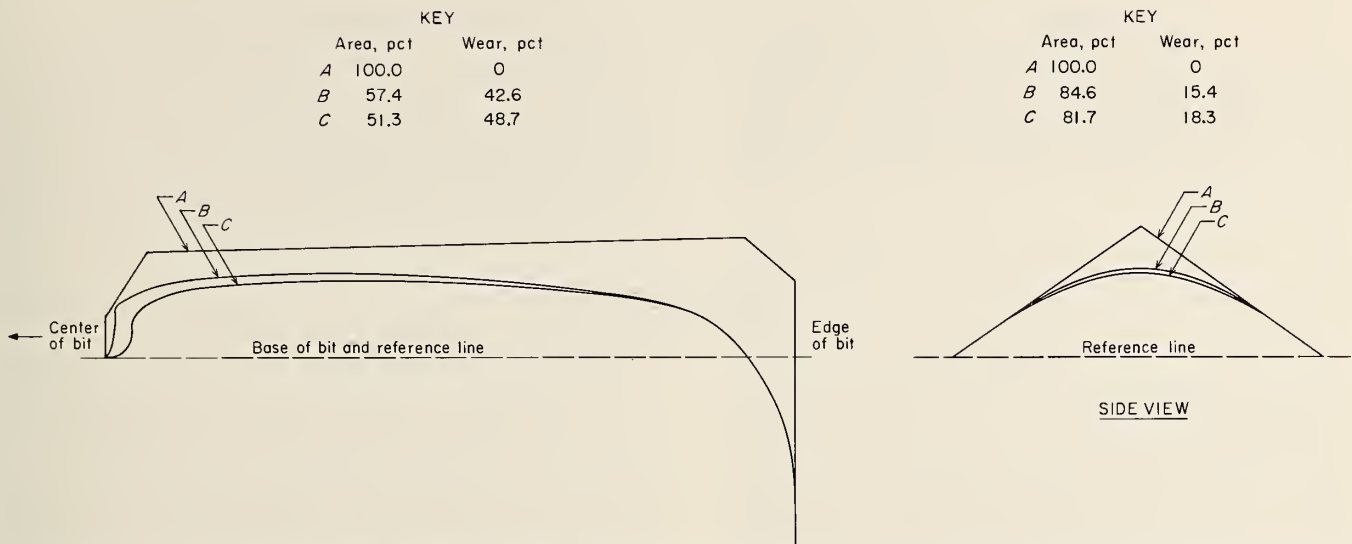


Figure 14.—Cross-sectional view showing optical comparator wear profile of typical 18-mm tungsten carbide wedge inserts from 51-mm cross bit.

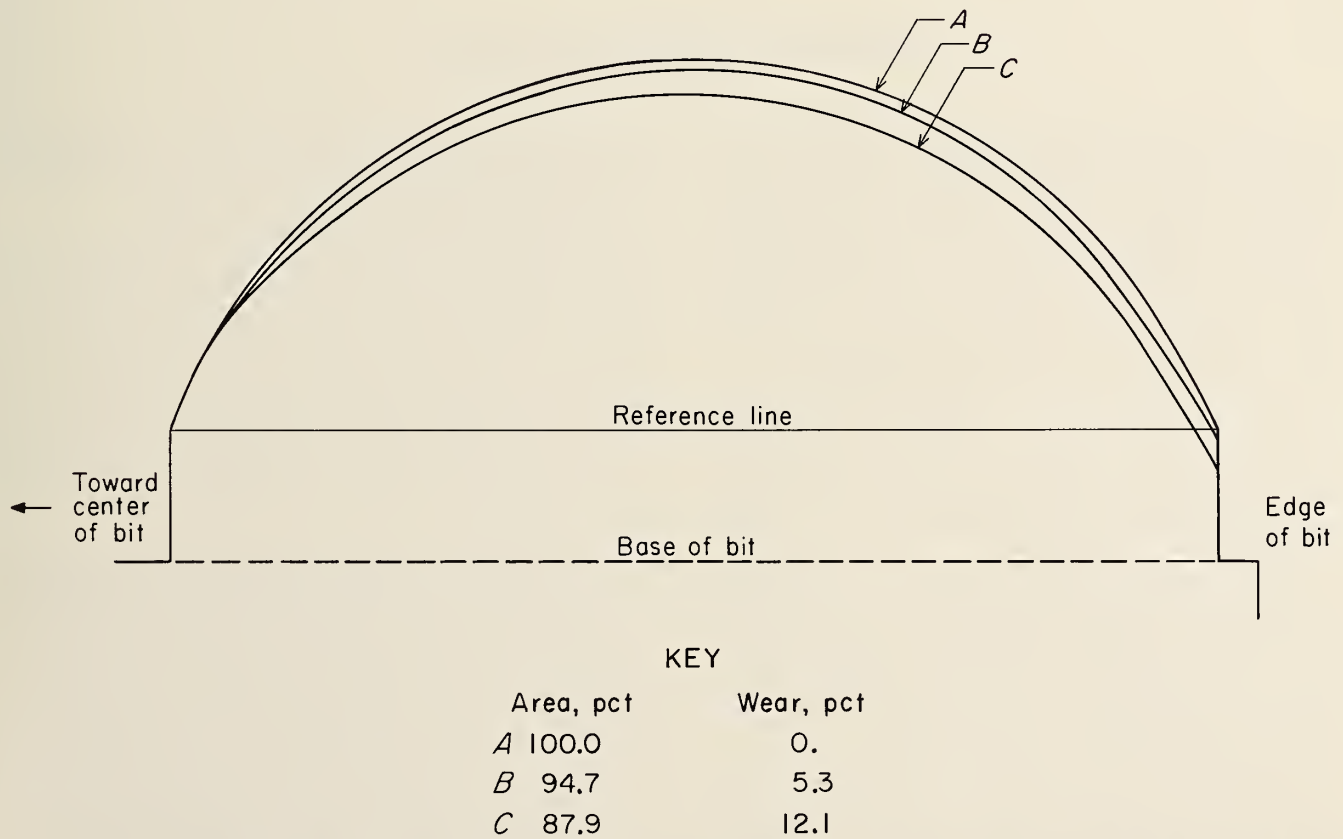


Figure 15.—Cross-sectional view showing optical comparator wear profile of typical 12-mm tungsten carbide buttons from 51-mm button bit.

POTENTIAL APPLICATIONS FOR IN SITU MINING TECHNOLOGY

Development of an in situ mining production field requires diamond core drilling to characterize an ore body and rotary drilling to create boreholes for injection and production. Thus, rock drilling is absolutely essential to in situ mining methodology.

It has been shown that PEO as a drilling fluid additive has improved drilling performance in a variety of diamond coring and rotary drilling applications. Since all of the rocks tested thus far (silicates, carbonates, oxides, and sulfide ores) can be treated by dilute PEO solution to achieve a ZSC condition, it would be a practical undertaking to incorporate this polymer into an in situ mining development plan as a drilling-fluid additive for all drills involved.

An example of an applicable in situ mining development is in the Santa Cruz ore deposit in Casa Grande, AZ, where the Bureau is conducting an in situ mining field demonstration project jointly with ASARCO Santa Cruz Inc. and Freeport Copper Co. The ore deposit being drilled is composed of a variety of copper oxide minerals hosted in the Oracle Granite and an adjacent quartz monzonite porphyry. This deposit is probably typical, as far as drilling for copper mineralization is concerned. The overall grade for the deposit is about 0.7 pct

copper; the deposit has never been worked by underground or surface mining, so it will be developed by surface drilling. Plans call for wire-line diamond drilling to obtain 2½-in core down to 2,000 ft; later these 4-in-diameter drill holes will be used as the injection wells. Recovery wells will be drilled by rotary drilling to the same depth, starting out as 12-in holes and tapering to 6-in holes at depth. PEO polymer solutions can be used in both instances, as the drills being used are normally equipped to handle fluids, and ZSC concentrations can be determined for each of the rock types encountered.⁶ Polymer concentrate can be added to the drilling water mixing system at the driller's discretion.

Other deposits that the Bureau may attempt to initiate for in situ mining research include cherty iron formations for manganese oxides and carbonates in the Cuyuna Range of Minnesota, the Duluth Gabbro formation for copper, nickel, and platinum-group-element sulfides, and a variety of volcanics, clastics, sediments, argillites, carbonates, and ferrous and siliceous rocks for gold and silver deposits. Laboratory results with PEO suggest that the polymer can be used confidently in drilling and should perform well with all rocks encountered.

SUMMARY AND CONCLUSIONS

Laboratory and field ZSC-controlled drilling tests have been conducted to determine the boundary conditions for the enhanced drilling phenomenon. Laboratory drilling tests were performed on Sioux Quartzite, Westerly Granite, Minnesota taconite, and Tennessee marble, using diamond-impregnated coring bits or tungsten carbide spade bits. Drilling fluids were prepared from chemical additives such as inorganic salts, cationic surfactants, acids, cationic polymers, or nonionic polymer in either DDIW, mine water, or tap water at concentrations below, at, and above the ZSC concentration.

Enhanced ZSC-controlled drilling performance was observed in laboratory drilling of high-silicate rocks, Sioux Quartzite and Westerly Granite; a medium-silicate rock, Minnesota taconite; and a nonsilicate rock, Tennessee marble; and in the field drilling of granite, taconite, and quartz-biotite schist, thereby establishing the applicability of ZSC-controlled drilling to a broad range of rocks.

The applicability of ZSC-controlled drilling using drag bit types was shown by the diamond-impregnated core drilling of Sioux Quartzite, Westerly Granite (5-6), and Minnesota taconite, and the tungsten carbide spade bit drilling of Tennessee marble. Field testing with percussive and rotary tricone and diamond-impregnated coring bits showed the applicability of ZSC-controlled drilling to these bit types.

Enhanced drilling performance has been obtained in the laboratory with solution concentrations of inorganic salts, cationic surfactants, cationic polymers, and nonionic polymer with some cationic character and in the field with PEO with some cationic character. All of these additives neutralized the rock surface charge, thereby establishing that the ZSC condition is the most important factor in ZSC-controlled drilling.

The applicability of ZSC-controlled drilling with respect to drilling-fluid water source or composition was established by achieving enhanced drilling performance with DDIW, mine water, and both plain and acidified tap water in the laboratory

and with mine waters in the field. Because of the increased amounts of anions in the mine and tap waters, more additive was required for surface charge neutralization in laboratory tests when using these waters in place of DDIW.

Maximum penetration improvements obtained for additives compared with baseline water penetration results in the laboratory ranged from 84 pct for Tennessee marble drilled with a near-ZSC concentration of aluminum chloride in DDIW to over 650 pct for Minnesota taconite drilled with a wide range of ZSC concentrations of PEO in mine water. Maximum bit life improvements obtained for additives compared with baseline water bit life results in the laboratory ranged from 56 pct for Sioux Quartzite drilled with a ZSC concentration of HTAB in DDIW to over 400 pct for Minnesota taconite drilled with a wide range of ZSC concentrations of PEO in mine water.

The laboratory tests demonstrated that PEO with partial cationic character (polarizable structure) in aqueous solutions was the best additive for accomplishing ZSC-controlled drilling. Unlike the cationic additives, which achieved the ZSC condition at only one concentration, PEO maintained the ZSC condition over a wide range of concentrations because of its polarizable structure. Thus, with PEO, the criticality of the solution concentration during drilling is greatly diminished. In addition, use of PEO resulted in much greater improvements in bit life and penetration than use of the cationic additives.

Initial field drilling results with ZSC-concentration solutions of PEO have also demonstrated enhanced drilling performance, compared with field drilling results with water

⁶The Bureau has developed a brief videotape and instruction sheet detailing the proper mixing procedure to get polymer into solution and in making dilutions. A copy of the video and instruction is available upon request from the Advanced Mining Division, Twin Cities Research Center, Bureau of Mines, Minneapolis, MN.

alone. In field diamond core drilling in granite, 2.3- to 3.1-fold improvements in bit life were obtained. In field percussive drilling of 100 ft of quartzite with cross bits, bit wear was slightly reduced. In field percussive drilling of 200 ft of quartz-biotite schist with button bits, bit wear was reduced by more than half. In field rotary tricone drilling in taconite, penetration rate was increased 69 pct (0.93 ft/min versus 0.55 ft/min) and bit life was increased 26 pct (3,400 ft versus 2,700 ft).

These initial field drilling results are encouraging but need to be validated by further field drilling tests.

However, development of the PEO polymer as a drilling fluid is far enough along that PEO can be used readily for exploration, quarry, blasthole, and production drilling, as well as in the development of an in situ mining well field where drilling in hard rock is a necessary aspect of mining.

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